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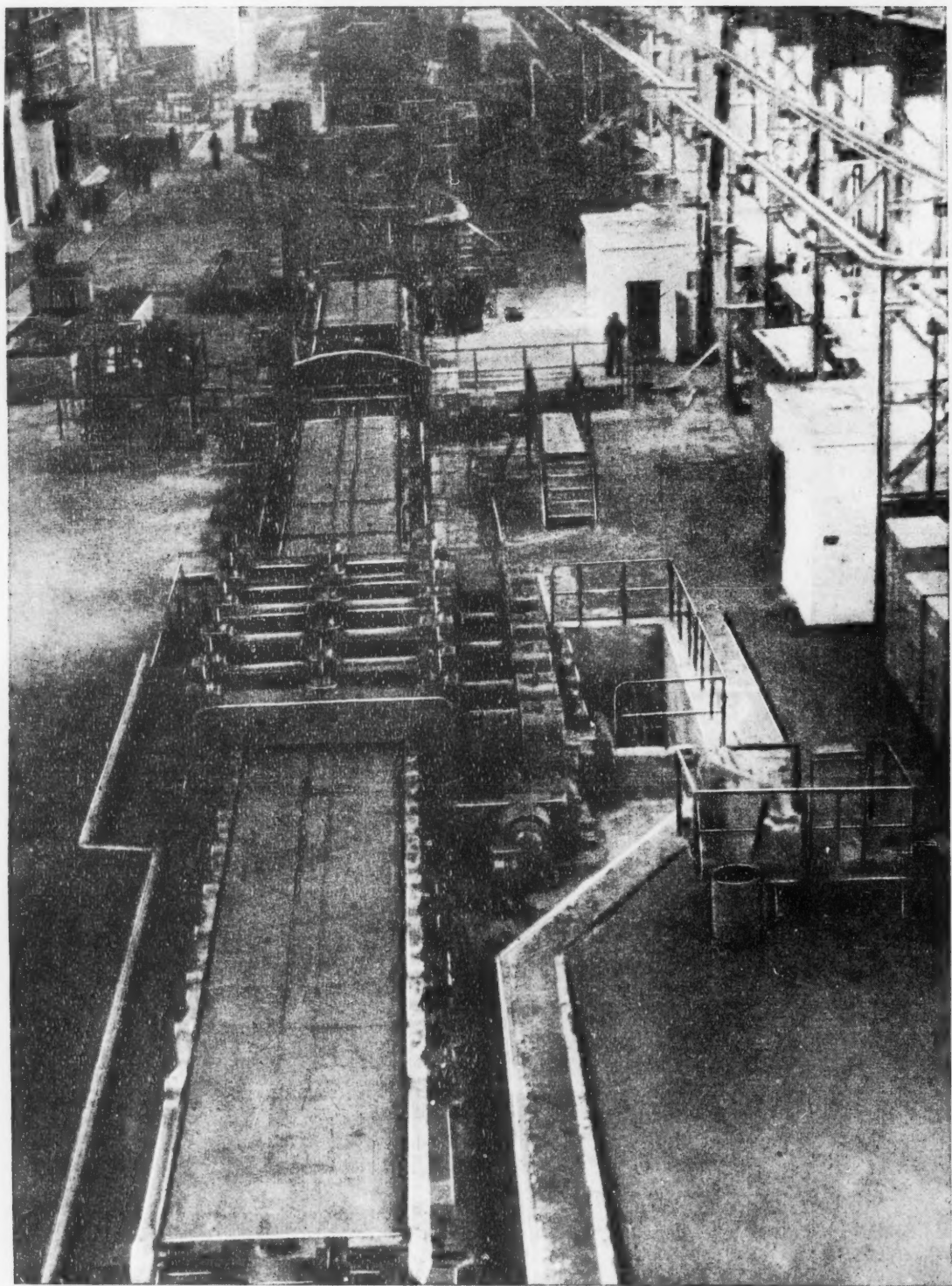
The Chelyabinsk Tube Works. Automatic plate preparation line in the electric welding plant.

ORGANIZATION OF PRODUCTION QUALITY CONTROL

(AS A MATTER FOR DISCUSSION)

The improvement of quality and the reduction of rejected products is a very important problem of the Socialist industry, and the operating personnel of the plants as well as the staff of the departments of technical control contributes to its solution.

The present arrangement of the OTK (Department of Technical Control) workers "educates" the operating personnel in such a way that the technical control staff is "chiefly responsible" and only after that is the operating personnel of the plants responsible for the quality of production. As a result of this erroneous attitude the OTK workers are dispersed along the technological production line and carry out the post-operational control in many plants and works. Hence, the very large number of OTK workers.

Post-operational control by the OTK relieves the operating personnel from the duty of a strict observance of the established methods of production and this fact is confirmed by the analysis of the operation of some plants of the "Serp and Molot" Works. For instance, the personnel of the OTK section in the rolling plant made 180 reports in 9 months of 1956 on the breaches of technological instructions caused by: upsetting of temperature conditions (30 reports); disregard of rules relating to metal charging into the furnace (43 reports); rolling of metal with worn-out or uncleaned rolls (52 reports); rolling of metal on worn-out housing (42 reports); poor stamping of metal (12 reports); disregard of the rules of metal cooling (1 report).

All this occurred in a plant where the senior welders, senior mill operators and foremen are highly skilled and have considerable operating experience and where there are special OTK inspectors for ingot charging, and at individual mills. The inspectors did not assist in eliminating the breaches (some of them resulted in faulty products) but only wrote out special warnings regarding the faults which had occurred in production. It should be mentioned that these warnings had no effect on the violators of the industrial process. After considering the 180 reports compiled by the OTK personnel, the management of the plant penalized the offenders in 14 cases only. The offenders in the remaining 166 cases were not penalized.

It shows that the plant workers do not contribute to an appropriate struggle for the quality of production and that the inspection of the OTK controllers is a mere formality.

The qualifications of many OTK workers performing post-operational control are, as a rule, below the qualifications of plant workers. The number of control personnel is excessively large. An analysis of the staff on several plants shows that the control posts on some sections and operations were established many years ago and at present are either absolutely superfluous or can be combined with some other posts.

In several places the control should be carried out by the plant operators.

Many OTK workers are not fully occupied. Frequently, some workers of the OTK duplicate the works of other OTK workers or of the operating personnel in the adjoining plant.

With this, the importance of the foremen—indirect organizers of production—is reduced, there develops a formalism and an unhealthy tendency to shift the responsibility for the quality of the production on to the staff of the department of technical control.

Indisputably, an overwhelming majority of plants in metallurgical works steadfastly, from year to year, reduce the output of poor quality and irregular products, but raising the plant workers' responsibility would speed up this process.

In the majority of establishments the technical control is organized on the plant (shop) basis—in each plant an independent OTK section. Sometimes, moreover, local plant "interests" do not coincide with state interests, as for instance the delivery of defective material to the next plant (shop) in the line of production (inside a works).

The staff of the department of technical control take care mainly to seeing that the products intended for outside consumers should comply with GOST specifications and with technological conditions. On the other hand, the billets and semi-products delivered to the plants of their own works is at times not adequately controlled and not in strict accordance with technical conditions. With such practices in metal supply to the manufacturing plants the struggle to reduce faulty production becomes difficult. Moreover, the responsibility of the operating and the control personnel for output of high-quality product declines.

The time has come to put firmly the question of more appropriate organization of technical control in metallurgical works. In our opinion this reorganization should be started without delay, but carried out in two stages: 1) to revise the duties of every OTK worker, to prepare plant workers with appropriate skill (or education) with a view to reducing the number of OTK workers employed in operational control, leaving them mainly in the control of finished products; 2) to take this reduced personnel of the department of technical control from under the authority of the works manager.

In the first stage, in order to improve the organization of control in the plant it is necessary to consolidate separate sections and to abolish the sections where there is no necessity for their existence.

The responsibility for the output of high-quality product is to be put indirectly on the heads of the operational sectors, heads of shifts, senior foremen and foremen, team leaders and skilled operators. Thus the services of the OTK personnel in the internal plant operations can be dispensed with. It is interesting that in many cases plant workers, realizing their increased responsibility, object to the removal of OTK personnel from the operational control.

The pay system should provide incentive for the operating personnel to improve the quality of production. When rolling mill operators of the plate rolling mill at the "Serp and Molot" Works began to receive a bonus according to the output of first grade steel, the output of first grade plate increased from 92.8 to 95.3%. In this mill there is no operational control by OTK.

The personnel of the department of technical control should check the quality of final products and semi-products only on their being transferred from one plant (shop) to another.

Regardless of the fact that the "Serp and Molot" Works, manufacturing a great variety of products, has smaller OTK staff than other works, we, nevertheless, regard the following measures advantageous:

1) to reduce the number of the OTK sections from 12 to 5 or 6; to establish sections of steel making, rolling and plating plants, outward inspection and transport-and-charge and repair shops; to reduce the number of controllers on technological operations;

2) to transfer the management of the control laboratories which at present belong to the OTK sections, to the main Works Laboratory;

3) to transfer all grading and sorting operations to the corresponding plants (shops).

These measures will make possible a substantial staff reduction in the department of technical control of the works.

It is necessary to devote more efforts to the development of perfected methods and means of control, as the methods now in use require a large number of controllers.

These measures as well as the engagement of the operating personnel in the everyday struggle for the improvement of quality in production will, undoubtedly, result in an improvement of quality and economy of production.

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N.P. Inozemtsev, Ya. K. Sokol, I.F. Rysev, D.A. Tarasenkov, S.I. Zamyatin.

("Serp and Molot" Works)

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BLAST FURNACE PRODUCTION

METHOD OF CONTROLLING THE OPERATING VARIABLES
IN THE BLAST FURNACE OPERATION

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In the operation of blast furnaces, especially on an unprepared charge with a fluctuating carry-out of the blast furnace dust, the amount of ore varies frequently. These variations have an effect on the thermal conditions in the furnace and cause a substantial fluctuation of the basicity of the slag and of the silicon and sulfur content in the pig iron.

With a view to reducing these fluctuations, a method has been developed in our works for correcting the ore loadings, limestone consumption and thermal condition of the furnace, the method being based on the calculations and operating indices of Blast Furnace No. 2 in the Novo-Tula Metallurgical Works.

The furnace produces casting metal from the charge consisting of 50% Kireev ore (brown hematite), 32% Krivorog ore and 18% KMA* sinter. The blast is oxygen-enriched, up to 26%, the blast temperature is 750°C and the humidity 40 g/m^3. The coke consumption is 1 - 1.05 $t/m^3/24$ hrs. The slag yield is 0.8 – 0.9 t/t of pig iron.

The changes of ore loading, limestone consumption and other operating variables have been related to the coke consumption per 1 t of metal. The following relationships between the variable operating parameters and the consumption of coke have been established:

1) A change of 85 kg/t in specific coke consumption causes a change of 1.0% in the silicon content of pig iron.

2) The change of ±0.1 in the optimum ore loading is equivalent to a change of ₊60 kg in the specific consumption of coke.

3) A change of ±100°C in the optimum blast temperature is equivalent to a change of ±60 kg in the specific consumption of coke.

4) A change of ±12.5 g/m^3 in the optimum blast humidity is equivalent to a change of ∓60 kg in the specific consumption of coke or to a change of ∓100°C in the blast temperature.

5) A deviation of 1.0% from the optimum value of the CO_2 content in the gas is equivalent to a change of ±30 kg of coke per 1 t of coke consumed.

6) A change of ±100 kg in the specific output of slag requires a change of ±50 g/t in the specific coke consumption.

7) A change of ±0.2 in the optimum ore loading, by means of removal or addition of an ore with 16−18% SiO_2 content, causes a change of 0.1 in the basicity of the slag $Ca:SiO_2$. With a decrease of ore load by 0.1, resulting from an increased (above average) dust carry-out (with mean SiO_2 content 22%), the basicity of the slag increased by 0.1. A change of ±1.4% in the silicon content of metal changes also the basicity of the slag by ±0.1. Such a change (±0.1) in the basicity of the slag is equivalent to a change of ±300 kg in the charge of limestone (with coke charge 4.4 t).

Using these data the foreman of a blast furnace can, in time, determine the significance of the changes taking place during the blast furnace process and take measures for regulating the furnace operation. For example, if the dust carry-out decreased or if, by mistake, the ore load increased by 0.03 − 0.1 above the optimum, the smelting of this charge requiring more than 2 - 3 hrs., the furnace foreman's task is: to determine optimum ore amount in subsequent charges delivered to the furnace; to determine the time when the charge with an increased

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* Kursk magnetic anomaly.

ore amount will enter the hearth and how much time is required for its smelting; to plan with what thermal factor the required additional heat for the charge with an increased ore load is to be compensated; to calculate how much the temperature of the blast should be increased or its humidity decreased in order to compensate for the deficient heat during the smelting of these charges and to enter on the report sheet the tasks for the next shift.

Let us assume in the first shift, from 1 - 3 o'clock, the ore loading in 18 deliveries was 0.1 above the optimum amount (increased from 1.65 to 1.75). Knowing the residence time of the materials in the furnace, the foreman will determine that the first delivery with the increased ore amount will enter the hearth (in the normal operation of the furnace) in 7 hrs. Hence, these charges will be smelting from 8 to 10 o'clock during the next shift. In order to compensate for the deficient heat for smelting this charge it is necessary, half an hour or an hour prior to its approaching the hearth, to increase the temperature of the blast and to maintain it at the increased level for two hours (from 7 to 9 o'clock), or to reduce the humidity of the blast. As 0.1 of the load is equivalent to 60 kg of coke per 1 ton of metal, and 100°C of blast temperature or 12.5 gm/m^3 of blast humidity are also equivalent to 60 kg of coke per 1 t of metal, it is necessary for the compensation of the deficient heat to maintain the blast at a temperature increased by 100°C for two hours, or raise the temperature by 50°C, and to reduce the humidity of the blast by 6.2 gm/m^3.

The foreman should take these measures starting at 7 o'clock or 7:30 during his shift, and enter into the report the tasks of the next shift for the maintenance of the operating conditions. The temperature should be raised and lowered gradually. After the two-hour operation of the furnace under the changed conditions the normal blast operation is restored.

When the ore loading is decreased the thermal conditions of the hearth are regulated by lowering the blast temperature or increasing its humidity.

When there are large deviations of ore load from the optimum amount, then, apart from the above measures, ore-free charges or charges with reduced ore amount should be introduced.

If on the same ore loads, limestone consumption and grade of produced metal, the utilization of the chemical gas energy deteriorates—which is easily determined by the controlling and measuring instruments and by the reduced CO_2 amount in the gas—then it is necessary to take measures for the equalization of the gas stream and for the maintenance of the thermal conditions in the furnace by changing the temperature or the humidity of the blast. For example, the instruments indicate the development of channeling and the drop of CO_2 from 12 to 11%. The foreman, knowing that the 1% drop of CO_2 content in the gas is equivalent to 30 kg reduction in the specific coke consumption and that it can be compensated by 50°C increase in the blast temperature or by 6.2 gm/m^3 reduction of humidity, has, in this case, to increase the temperature of the blast or reduce the humidity of the blast correspondingly. When the gases begin to be utilized normally and the CO_2 content attains the optimum value, then the introduced changes in the blast should be cancelled and the normal blast conditions restored.

It is also necessary to adjust ore loads and limestone consumption when a switch-over from making of one metal grade to another is made. Then the heat losses due to silicon reduction and the change in the amount of formed slag are taken into account. For instance, if the furnace is switched from producing LK-00 metal to LK-1 metal the silicon content will be decreased by 1%. In this connection, the amount of slag will increase by 50 kg per 1 t of metal at the same basicity, and by the decrease of silicon content in the metal by 1%, 85 kg of coke for each ton of metal will become available. Taking into account the additional slag and the available surplus of coke it is necessary to adjust the ore charge and limestone consumption.

The amount of slag increased by 50 kg, due to the decreased intake of silicon by the metal, necessitates an additional specific coke consumption of 25 kg/t. Hence the available surplus of coke will be 85−25 = 60 kg/t. The ore charge would have to be increased by this quantity but at the same time the heat losses on the slag from the additional ore must be taken into account. As 0.1 of ore charge is equivalent to 60 kg of coke per 1 t of metal, the increase of ore charge by 0.1 will result in an increase of slag output by 40 kg at the maintained basicity and this will require an additional 20 kg of coke. Therefore, on changing over from making LK-00 metal to LK-1 metal, the final quantity by which the ore charge has to be increased will be only 60−20 = 40 kg of coke. The additional amount of ore charge required is determined by the following ratios:

$$60 \text{ kg of coke } - \quad 0.1 \text{ charge}$$
$$40 \text{ kg of coke } - \quad \text{x charge}$$
$$x = \frac{40 \times 0.1}{60} = 0.07 \text{ charge.}$$

If the optimum charge for metal LK-00 was 1.63, the charge for metal LK-1 will be $1.63 + 0.07 = 1.7$ t of ore/t of coke.

In addition, the limestone consumption must be adjusted. For the slagging of surplus silica, because of decreased intake of the silicon by the metal and the increased transfer of SiO_2 into slag, the following is required:

$$1.4\% \text{ Si } - \quad 300 \text{ kg limestone per delivery}$$
$$1.0\% \text{ Si } - \quad \text{x kg}$$
$$x = \frac{300 \times 1}{1.4} \approx 200 \text{ kg of limestone per delivery.}$$

For the slagging of SiO_2 in the additional ore (from the increased ore charge by 0.07) the limestone requirements are:

$$0.2 \text{ charge } - \quad 300 \text{ kg}$$
$$0.07 \text{ charge } - \quad \text{x kg}$$
$$x = \frac{300 \times 0.07}{0.2} \approx 100 \text{ kg of limestone per delivery.}$$

Total requirement of additional limestone on changing from production LK-00 metal to LK-1 metal is $200 + 100 = 300$ kg.

In the case of increased or decreased ore loads, depending on the changes of physical properties of the raw material (difficultly reducible lump ore or easily reducible normal size ore), it is also necessary to make some adjustment in the limestone consumption. On a decrease in ore charge by, let us say, 0.07, it is necessary to lower the limestone consumption by –

$$0.2 \text{ charge } - \quad 300 \text{ kg}$$
$$0.07 \text{ charge } - \quad \text{x}$$
$$x = \frac{300 \times 0.7}{0.2} = 100 \text{ kg.}$$

On an increased ore charge the consumption of the limestone is raised accordingly.

If the ore charge change occurred in a few deliveries only and its effects were immediately corrected, the limestone quantity in the subsequent deliveries should not be changed as such changes would result in a large variation in the basicity of the slag and consequently a poor operation of the furnace.

The variation in SiO_2 content in Kireev ores amounts to $\pm 3.7\%$; in grade 23, Krivorog ore, to $\pm 2.3\%$; and in the KMA sinter to $\pm 0.43\%$. Assuming the upper limit of SiO_2 content in all the ores simultaneously, the basicity of the slag would decrease by 0.15. Therefore, it is necessary to adjust the limestone consumption also when a sharp change of SiO_2 content in all or in one kind of ore occurs.

If in grade I, Kireev ore, the SiO_2 content increases by 3% above the mean value with the ore consumption of 1700 kg, the input of SiO_2 increased by $1700 \times 0.83 \times 0.03 = 42$ kg. For slagging this amount of SiO_2 the following is required:

$$\frac{42 \times 1.2 \times 1.05}{0.53} \approx 100 \text{ kg of limestone.}$$

On SO_3 content change in limestone up to 1%, or change of ash content of coke by 1%, the basicity change of slag is 0.01 and 0.02 respectively and is practically insignificant. In this case the limestone consumption need not be corrected. More pronounced changes of SiO_2 content in limestone and of ash content in coke should be taken into account and a corresponding adjustment of limestone consumption should be made. When there is a combination of several factors simultaneously affecting the basicity of slag, the limestone consumption should also be adjusted.

By testing the actual changes of ore charges, limestone consumption and corresponding changes of silicon content in the metal and basicity of the slag as well as the effect of changes in the temperature and humidity of the blast on the thermal conditions of blast furnace No. 2, a good agreement with the above considered relations has been established. It indicates the expediency of adopting this method for the practical control of blast furnace operation. Specific conditions in each furnace, the quality of raw material, and the grade of metal produced should then be taken into account.

CONTROL OF BLAST FURNACE OPERATION ACCORDING TO
STATIC PRESSURE DIFFERENCES

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In recent years on some blast furnaces in the USSR, instruments have been installed for measuring the resistance which has to be overcome by the gases in their passage through the stock of charge materials from the tuyere zone to the top of the furnace.

By determining the total static pressure drop between the hot blast and the top, as well as pressure drop on separate sections of the charge (hot blast— middle of the charge, middle of the charge — top), the change in the gas permeability of the charge in these sections can be detected in time and not only the places where the seat of the charge hanging (scaffolding) originates, but also the probability of hanging, can be determined.

Therefore, the instruments determining pressure drops constitute an efficient means of controlling the blast furnace operation. The practical application of such equipment in several works, in particular in the Kuznetsk Metallurgical Combine, confirms this fact.

The control of statical pressures, however, can give good results only if an absolute reliability of measurements is ensured. In this respect, it is convenient to measure the total pressure drop (hot blast — top), but the
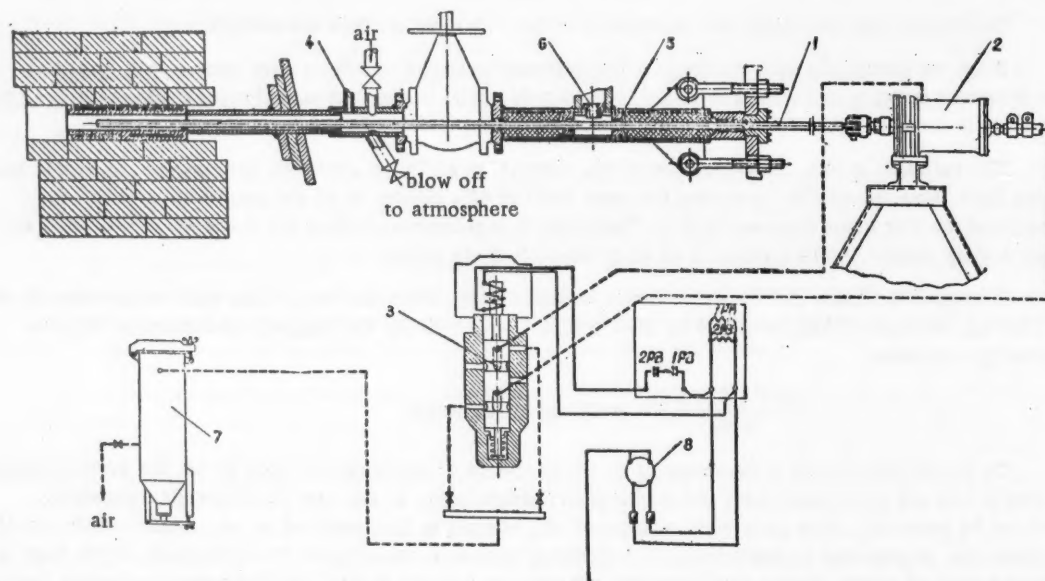

Fig. 1. Equipment for clearing the air-intake tube. 1) bar; 2) servomotor; 3) solenoid valve; 4) gas take-off; 5) stuffing box; 6) opening for the delivery of lubricant to stuffing box; 7) air filter; 8) switch; 1 PB and 2 PB-time relays.

reading obtained is not very useful as it does not allow an accurate determination of the section where the change in the resistance of the charge to the passage of gases occurs. It is, however, essential for correcting the furnace operation to know exactly where these resistance changes originate — in the lower, middle or upper part of the charge. Depending on the location of an increased resistance formation, appropriate measures are taken to correct the furnace operation. Hence, it is necessary to measure static pressure in the middle section of the charge.

The equipment for air intake in the middle part of the charge operates under very difficult conditions: the temperature in this zone reaches 1000°C, gas pressure is 12,000 mm water gage, forming slags and coke fines block the tube outlet.

The equipment for air intake on the KMK (Kuznetsk Metallurgical Combine) has been in continuous operation for over a year. The blast furnace personnel utilizes the readings of the instruments extensively for the control of the blast furnace operation. This has been achieved by the adoption of an automatically operated system of mechanical clearing of the air inlet equipment in the middle section of the furnace charge.

The inlet equipment consists of a special cooler with 75 mm diameter opening (Fig. 1). The tube is cleared automatically with a special bar actuated by compressed air.

The sequence of the equipment operation is as follows. Every 30 minutes relay 1 PB switches on relay 2 PB, which sends electric current to the winding of a solenoid valve. Under the action of the magnetic force of the solenoid the valve opens and air passes through the valve to a servomotor of 500 mm stroke length. The piston of the servomotor moves the bar forward to clear the gas inlet. After the switching off of the solenoid, the valve, under the action of a spring, goes down, the air enters the other part of the servomotor and, by the pressure on the piston, moves the bar back into the initial position. The cycle is repeated after 25 seconds. After 30 minutes the relay again switches on the solenoid twice and the bar again clears the inlet. In the extreme left position the bar penetrates the furnace to the extent of 50 - 80 mm. The clearing system is reliable in operation and does not require much maintenance.

The static pressure drops on the section hot blast — middle of the charge, and the middle of the charge — top, are measured by means of two differential manometers of DPES model and one double electronic bridge, which constitute a combined system of two electronic bridges mounted in one housing and working on one common diagram tape. The recording gives a graphic representation of the extent and character of the changes in static pressure drops.

The adoption of the gas inlet clearing system tested on the Kuznetsk Metallurgical Combine, and the recording of the pressure readings, makes possible a considerable improvement in blast furnace operation control and the automation of some control operations: gas pressure at the top of the furnace, blast delivery, its temperature and humidity. Samples of pressure drop recordings are shown in Fig. 2.
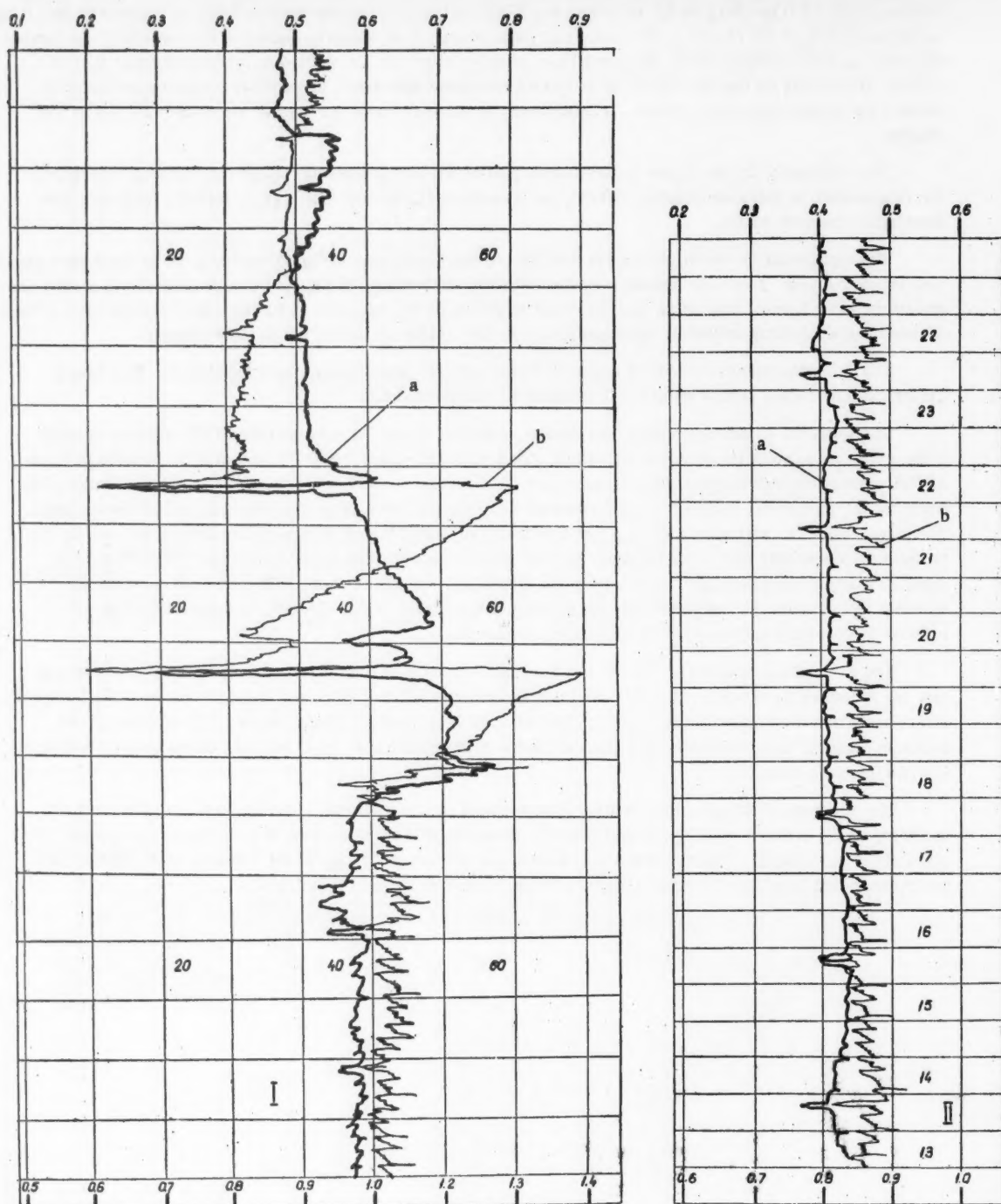
Fig. 2. Recordings of static pressure drop: a) pressure drop on the section hot blast— middle of the charge;
b) section middle of the charge —furnace top;   I) furnace operating with scaffoldings;   II) regular furnace
operation.

HIGH TEMPERATURE HOT BLAST STOVE

(Reported at the All-Union Blast Furnace Conference)

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(Gipromez)

Blast temperature may be raised by different methods. It is possible to increase the gas combustion temperature under the dome of a stove, to increase the consumption of gas and air for combustion, i.e., the temperature of the waste gas, and to construct stoves with a large heating surface. Not all these methods, however, are equally effective.

Calculations showed that if the gas combustion temperature under the stove dome is raised by 350°C, it is possible to raise the hot blast temperature by 300°C and an increase of 500°C (from 200 to 700°C) of the waste gas temperature enables the blast temperature to be increased by 250°C (from 850 to 1100°C) while a more than double increase in the heating surface of stoves enables blast temperature to be raised by only 70°C. This comparison shows that the latter method is the least effective. Secondly, although it does ensure an increase in blast temperature up to 1100°C, it does not guarantee normal continuous operation of the installations below the checkers and it requires an enormous consumption of blast furnace gas for heating the blast. The first method, however, if suitable refractory materials are available, enables such a temperature to be maintained, makes possible an increase in blast temperature to the requisite limits and ensures good thermal efficiency. This method is the most effective.

Secondly it is essential to find a sufficiently refractory material for lining the combustion chambers, the dome and for the upper courses of the checkers (which heat up to a temperature of more than 1100°C).

A suitable refractory is a high alumina brick containing 45 - 46% alumina which is at present manufactured by our refractories industry. The refractoriness of this brick is 1780°C, temperature of initial deformation under a load of 2 kg/cm^2 is 1500°C.

The gas combustion temperature is raised by an addition of 8 - 12% of coke oven gas to the blast furnace gas.

The basic operational characteristics of the stove with a heating surface of 30,000 sq m, obtained from thermodynamic calculations are given below:
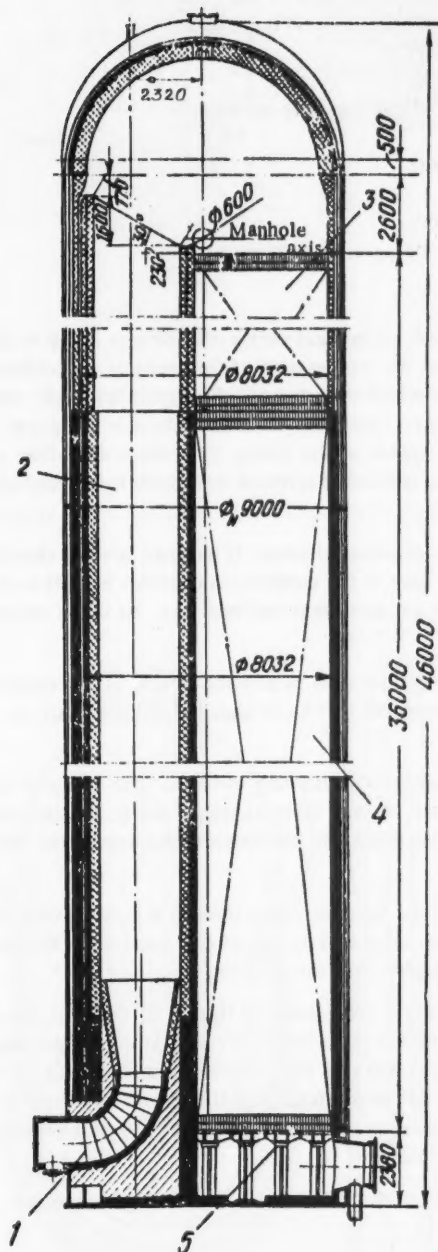

Fig. 1. Section of high temperature hot blast stove: 1) burner; 2) combustion chamber; 3) high Alumina brick checkers 9 m high; 4) firebrick checkers 27 m high 5) heat resisting cast iron installations beneath checkers.

Blast consumption:	3,500 cu m NTP per min.
Temperature of cold blast:	150°C
Period on gas:	1.9 hours
Period on blast:	1.0 hour.
Calorific value of the gas:	1240 cals per cu m at NTP
Gas consumption:	
Coke oven gas:	11,000 cu m at N TP per hour
Blast furnace gas:	73,000 cu m at NTP per hour
Blast temperature:	1250°C
Waste gas temperature:	300°C
Coefficient of heat exchange:	14.98 cal per hour per sq m
Drop in blast temperature for the period:	240°C
Rise in waste gas temperature for the period:	150°C
Productivity of gas burners:	55,000 cu m at NTP per hour

A section of the high temperature stove is shown in Fig. 1.

The dome of the stove is in the form of a semisphere with an internal radius of 3750 mm and it is lined with a single course of high alumina brick 450 mm thick. Since the temperature of the products of combustion is higher than normal, dome insulation is increased: between the high alumina and the tripoli brick 123 mm thick is a layer of light weight brick 1113 mm. Between the dome lining and the jacket there is left a gap of about 500 mm in a vertical direction to allow for temperature growth of the lining. To prevent any effect of the thrust of the dome on the stove jacket the base of the dome is fixed to a metal rim which is not connected with the jacket.

In order to reduce wear of the dome, the mouth of the combustion chamber is inclined toward checkers at an angle of 30°. This increases the discharge cross sectional area of the combustion chamber by 35% so that the products of combustion are discharged at a lower speed and are somewhat inclined from the dome toward the checkers.

In the high temperature stove the heat input is markedly greater than in existing stoves. In this connection the active cross sectional area of the combustion chamber is increased to 6 s q m and, in the upper part, to 7 sq m, i.e., by 40% compared with normal stoves.

The combustion chamber is lined with two courses of brick without jointing material. The external course is 230 mm thick and consists, in the upper high temperature zone, of high alumina brick, and in the remainder, of fireclay stove brick. The inside course consists of high alumina brick 230 mm thick in the upper zone and 345 mm in the remainder.

A gas- air mixture is fed to the combustion chamber from the bottom, passes through a smooth bend and opens out (diffuses) in the direction of the chamber. The mixture is ignited by means of a special device for ignition and control of combustion which is located somewhat higher than the diffuser.

In the upper high temperature zone of the stove wall there are two courses of lining: an external one of light weight brick 113 mm thick and an internal one of high alumina brick 230 mm thick. In the region beneath the checkers the walls are lined with two courses of firebrick each 230 mm thick and each close up against the jacket. The remaining lining consists of one course of firebrick 345 mm thick. Tripoli brick 65 mm thick is used close up against the jacket from the region below the checkers to the dome. The gap of 60 mm between the lining of the tripoli brick is rammed with fireclay asbestos filling.

The jacket of the high temperature stove is no different from that of the hot blast stove of a standard blast furnace.

The checkers of the hot blast stove are single-stage with staggered compartments 45 X 45 and 35 X 35 mm of shaped brick. The gases and air can penetrate through the horizontal passages into the neighboring compartments. The upper courses of the checkers at a height of 9 m are lined with high alumina brick and the remaining courses with firebrick. Cross sectional area of the checker chamber is 37.3 sq m.
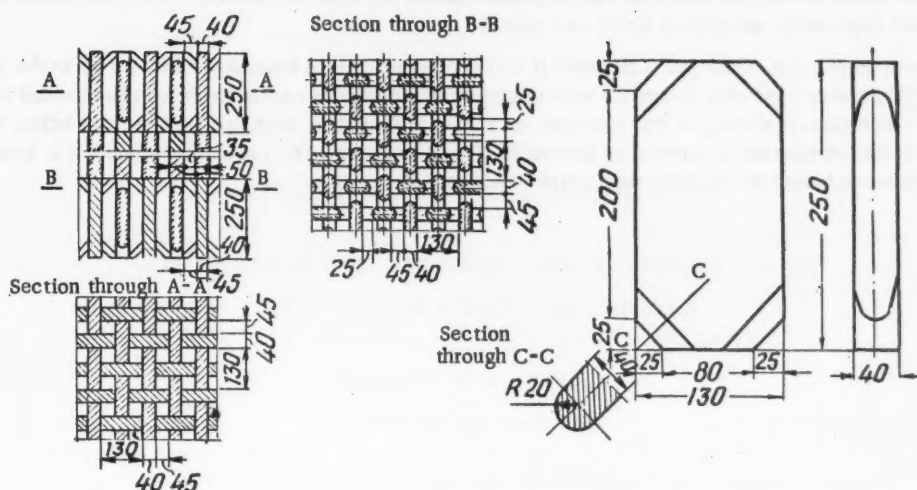
Fig. 2. Checkers of a high temperature hot blast stove: On the left — checkers. On the right — checker brick.

The design of the hot blast main and the tuyere device does not differ from the normal. The internal sluiceway of the hot blast main is lined with firebrick and the external sluiceway with light weight firebrick.

The connection between the blast main and the hot blast branch pipes is lined with concrete, with fireclay or high alumina filler on alumina cement. The tuyere sleeve is lined with firebrick. The nozzle is lined with molded fireclay rings with a wall thickness of 20 mm and length of ring of 350 mm.

A new design of hot blast valve has been developed for the high temperature stove (Fig. 3). This type of damper can operate at blast temperatures of up to 1450°C and at pressures of up to 3 kg per cm^2.

The body of the slide is cast without a horizontal joint. The slide itself can be made of cast or stamped parts. Inside the slide there are two spiral baffles by means of which a high speed is imparted to the water in the lower and peripheral parts, thus preventing the deposition of suspended matter.

The rings of the slide are made of refractory brick containing 75% alumina. They protect the body of the valve from sudden variations in temperature.

The gas burner must be modernized for the high temperature stove.

The existing IZTM burner with a productivity of 48,000 cu m of air and a pressure of 300 mm water column cannot ensure the heating of the blast up to 12,000°C for a blast furnace with an effective volume of 1513 cu m. Consequently it is suggested that there should be a centralized feed for the air for combustion
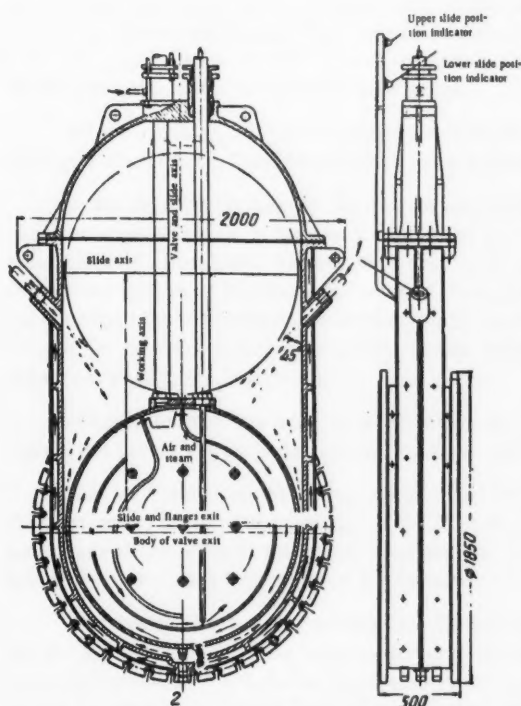

Fig. 3. Hot blast valve: 1) and 2) points at which the cold blast enters for cooling the valve.

instead of a general fan installation, consisting of two powerful fans each capable of producing 150,000 cu m of air at 480 mm water column per hour (one fan is operational and the other on reserve). Such installations which serve three hot blast stoves are already being used abroad.

Optimum combustion of the gas is obtained if it is completely mixed beforehand with air up to the point of ignition. This makes it possible to operate with minimum coefficient of excess air for combustion and with a short flame. Gas burners operating on this principle are known as flameless. With this type of combustion, thermal stresses in the combustion chamber may amount to 20 to 40 million cals per cu m per hour. It is intended to make a flameless burner for the high temperature hot blast stove.

STEEL MELTING PRODUCTION

RECONSTRUCTION OF OPEN-HEARTH FURNACES AT THE "KRASNY OKTYABR" WORKS

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The open-hearth furnaces of the "Krasny Oktyabr" metallurgical works in Stalingrad have a number of constructional features in which they differ substantially from modern furnaces. This is connected with peculiar features in the historical development of the works, founded in 1897.

The original works had one open-hearth steel plant (now plant No. 2) with 5-ton and 25-ton furnaces, constructed to French design. The period from 1927 to 1930 saw the building of open-hearth plant No. 1, with 75-ton furnaces designed at the works on the pattern of the furnaces of the old plant. In the course of the succeeding years, all the furnaces were rebuilt on a number of occasions, mainly with a view to increasing the capacity, which before the war was raised respectively to 25, 50 and 110 tons. During the war, the open-hearth plant of the works was largely destroyed and all the furnaces were put out of action. During the rebuilding period, no substantial changes were introduced into the construction of the furnaces. Thus by 1953, the works had three types of open-hearth furnace, excluding a new 130-ton furnace. In addition to differences in capacity, the furnaces also had constructional differences.

All the furnaces, with the exception of three 50-ton furnaces, which have partly been converted for natural gas, are oil fired. Compressed air under a pressure of 5 to 6 atm. is used for atomization.

In the constant increase in the furnace capacity, formerly effected by deepening and lengthening the bath, the upper structure of the furnace has been the main subject of modification. There has thus come about a considerable discrepancy between the capacity of the furnace and such important parameters as the area of the hearth, volume of the checkers, volume of slag pockets, cross section of vertical flues, etc. The consequence has been high fuel consumption and short life of various elements of the furnace (checkers and brickwork of vertical flues). The capacity of the slag pockets, especially after conversion of the furnaces to basic roofs, turned out to be quite inadequate.

Reconstruction of the open-hearth furnaces was commenced in 1953. All the reconstruction work was carried out during the projected general overhaul repairs (18—22 days) under operating conditions of the plant.

Fundamentally, the reconstruction provided for increasing the maximum thermal capacity by eliminating the discrepancy between the tonnage of the furnace and its principal parameters; modernization of the reversing equipment of the furnace, instrumentation and automatic control equipment; and increase in the rigidity and durability of the steel framework of the furnace.

The first to be reconstructed were the 25-ton furnaces No. 0 and 00. During the general repair, by widening the working space and the lower structure of the furnace on the casting bay side, the width of the bath was increased by 400 mm and that of the regenerators and slag pockets by 1000 mm. This made it possible to increase the area of the hearth from 13.3 to 15.6 m^2, the volume of the checkers from 40 to 54 m^3 and the total volume of the slag pockets from 16 to 26 m^3. A steel construction of increased rigidity and strength was provided for the working space and lower structure of the furnaces.

The measures adopted had an immediate effect on the working of the furnaces: the life of the checkers increased from 250–260 to 500 heats, the thermal capacity of the furnace increased, the daily melt of steel increased by about 17%, with a reduction in the specific fuel consumption of 5%. Due to the increase in productivity and length of campaign, furnace No. 00 in the first year following reconstruction produced an additional 4000 tons of steel.

Reconstruction of 50-ton Furnaces

The 50-ton open-hearth furnaces are situated in a building with columns spaced 5.5 meters apart and each occupies three spans with a total length of 16.5 meters. The working space (furnace proper) of the furnaces is 14.3 meters long and the total distance between the ends of the brickwork is 2.2 meters. The furnaces are located entirely in the furnace bay, the width of which (15 meters) is likewise inadequate, especially when it is remembered that the filled charging machines and box trains are also situated there. The furnace regenerators are situated beneath the working space.

Before reconstruction, the working space was supported on the masonry of the lower structure, and therefore the side walls of the regenerators were 850 mm thick (500 mm of supporting masonry of ordinary brick and 350 mm of firebrick courses). The regenerators and slag pockets had no casing and were not thermally insulated. The framework of the lower structure and working space consisting of a system of buckstays and horizontal girders, connected by tie rods, did not ensure the necessary rigidity. The upper structure of the furnace became considerably distorted during operation and the supporting walls rapidly broke down and became unstable. This complicated the execution of repairs to the regenerator chambers.

In view of the crowded disposition of the furnaces in the plant, reconstruction had to be carried out as far as possible without altering the external dimensions of the furnaces. The problem was solved by providing a steel supporting construction for the working space, and this steel construction at the same time also forms the framework for the regenerators.

With such a construction for the support of the working space, there was no need to construct masonry supporting walls, and the total thickness of the masonry of the side walls of the regenerators was reduced to 580 mm (including 115 mm of thermal insulation). It was thus possible to increase the width of the regenerator chambers and slag pockets by 550 mm. The masonry of the lower structure is enclosed in a welded casing of 8 mm thick steel plate.

Working Data of the Furnaces (weighted averages)

Working period	Duration Campaign, heats		Length of heats, hours – minutes				Mean weight melt tons	Metal melted per campaign, tons	Output per 24 hrs., tons		Kind of steel melted, %		
	Roof	Checkers	Total	Comprising					Calendar	Nominal	Rimming	Killed, carbon	Alloy
				Charging	Melting	Boiling							
Before reconstruction* . . .	441	441	7—39	3—07	2—54	1—38	49.0	21639	139.0	145.0	100	—	—
After reconstruction** . . .	503	503	6—41	2—38	2—31	1—32	48.5	24423	151.0	161.0	100	—	—
Before reconstruction* . . .	368	368	9—19	3—29	3—20	2—10	101.5	37286	238.0	251.4	2.0	70.6	27.4
After reconstruction** . . .	444	444	9—30	3—36	3—27	2—07	125.1	55425	294.0	303.0	—	58.0	42.0

* For three campaigns
** For four campaigns

As a result of the reconstruction, the volume of the checkers has been increased from 89 to 100 cubic meters, and the total volume of the slag pockets from 34 to 60 cubic meters.

The parameters of the working space have been altered. Improvements in the construction of the supports for the banks and a reduction in the thickness of the brickwork for the banks have resulted in an increase in

the area of the hearth from 18.6 to 20.6 sq. meters. An increase in the overhang of the back casing from 500 to 700 mm has enabled the slope of the back wall to be carried to the base of the roof, which has made fettling conditions much easier. The construction of the furnace end blocks (ports) has been altered and the arch above the mixing chambers has been raised by 400 mm. The cross section of the flame ports has been increased from 2.6 to 3.3 sq. meters. A modern type of rigid framework for the working space has been provided on the reconstructed furnaces. The "Simplex" reversing valves with a through cross section of 1.2 × 1.2 meters have been replaced by "Hay" slide valves with a cross section of 1.2 × 1.4 meters offering much less resistance.

Due to the widening of a number of cross sections of No. 4 furnace, the volume of the checkers and slag pockets has been increased, and improved construction of the ports has considerably increased the heat capacity of the furnace. The average length of a heat has been reduced by 58 minutes (see table), mainly due to a reduction in the charging and melting periods; the increase in productivity for a nominal 24 hours has been 11%; the working campaign of the furnace has been increased by 14%.

Reconstruction of the 110-ton Furnaces.

The 110-ton open-hearth furnaces are situated in the furnace bay of the building with a distance of 12.5 meters between columns; the distance between the furnace axes is 25 meters. The working space has a length of 19.12 meters, so that the distance between the end walls of adjacent furnaces was equal to 5.88 meters. Before the reconstruction, the hearth was supported on a rigid horizontal frame consisting of three longitudinal beams and six transverse beams (Fig. 1). The frame was mounted on ten columns, four on the back of the furnace, five on the front and one between the regenerators. In addition to vertical regenerators, the furnace has horizontal regenerator chambers with double checkers. The horizontal regenerators pass under the charging stage below the level of the plant floor. A narrow gage track for removing slag is laid on a ferroconcrete covering above the chambers.

The essentials of the work done and its results may be gathered from the example of No. 8 furnace. The reconstruction aimed at increasing the capacity of the furnace to 130 tons, the basic parameters of the furnace at the same time being brought into line with the new capacity.

After dismantling the furnace, the column between the regenerators was removed; the other columns were shortened by 1000 mm. Fresh supports for the beams were welded to the columns and on them was mounted a new supporting construction, consisting of two longitudinal bearers of a height of 1000 and 1300 mm, rigidly connected to the furnace ends. The top surface of the bearers is 500 mm lower than the original frame. Welded on the supporting structure are two types of transverse bottom beams: buckstay channel beams, 500 mm in height, rigidly connected to the buckstays of the hearth, and intermediate structural beams, 800 mm in height at the center. The level of the hearth floor has been lowered by 360 mm, which has made it possible to increase considerably the brickwork of the bottom and in particular, the thickness of the fettling from 80 to 250 mm. Thus, with the new construction of the bottom, the capacity of the furnace can be still further increased by reducing the thickness of the furnace bottom brickwork (Fig. 2). At the same time, there is less risk of metal breaking out through the bottom (this has happened formerly).

(hatched)	magnesite
(hatched)	magnesite – chromite
(hatched)	Dinas
(hatched)	fireclay
(hatched)	ordinary brick
(hatched)	heat-insulating brick

Arbitrary designation of refractory materials used in the figures.

A relatively arbitrary distribution of the furnaces along the plant has made it possible to increase the length of the hearth to 21 meters. Simultaneously with the reduction in thickness of the banks, this has made it possible to increase the length of the bath from 10.4 to 12 meters. Due to a more rational construction of the casing of the back wall and full utilization of the available space of the plant, the width of the working space has been increased and the bath widened by 400 mm. Thus, the most important parameter of the furnace, the area of the hearth, has been increased from 39.3 to 51.0 sq. meters. The volume of the bath, necessary for accommodating a 130-ton charge, has been obtained with an insignificant increase in depth (from 820 to 850 mm).

The increase in the overhang of the back wall casing by 200 mm has made it possible to incline the back wall as far as the abutment beams, with an angle of slope along the brickwork of about 55°. As result of the
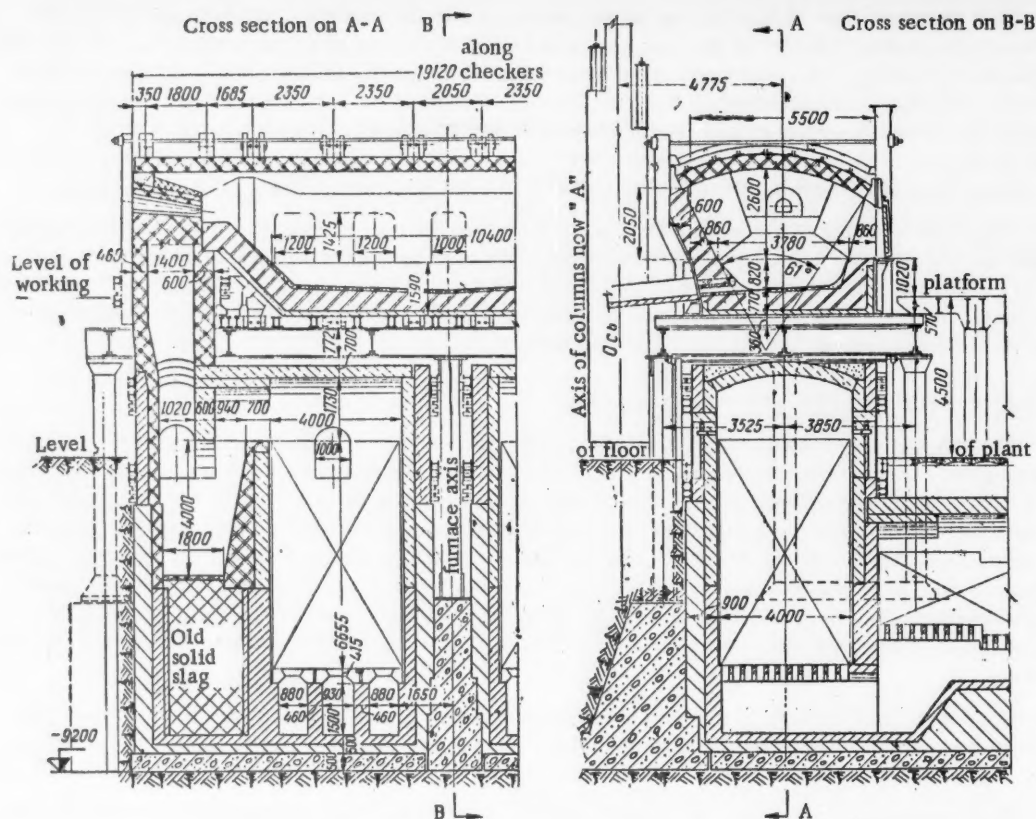
Fig. 1. Longitudinal section cross section of 110-ton open-hearth furnace before reconstruction.

reconstruction, the back skewback channel has been considerably lowered, so that the central angle of the roof has been increased from 61 to 73°. The furnace has been provided with a braced suspended roof with controlled thrust arches (system of the Steel Planning Department).

The construction of the ports has been improved and their dimensions have been approximated to those recommended for Venturi type ports. The arch above the combustion chambers has been raised by 600 mm. Burners with an adjustable angle of inclination have been mounted in water-evaporating coolers in the brick-work of the ports.

The cross section of the double vertical flues has been increased from 4.2 to 6.24 sq. meters, mainly because the ends of the furnace have been moved apart.

Before reconstruction, the spacing of the axes of the buckstays was unequal, the result of which was that the middle charging door was 200 mm narrower than the others. This gave rise to difficulty in operating the furnace; for instance, there had to be two types of skewback channels, frames and doors for the charging door-ways, and the width of the middle doorway set a limit to the size of the charging boxes. After reconstruction, these drawbacks, were eliminated and all the doorways now have the same width (1200 mm).

By improved utilization of the accommodation below the working space and lengthening the furnace, the capacity of the regenerator chambers and slag pockets has been increased. An original framework has been provided for the furnace bottom, comprising two rigid frames embracing the lower structure: one at the level of the feet of the regenerator arches and the other below, at a depth of 1200 mm. Along the furnace axis, be-tween the regenerators, each frame has a cross-piece of welded H-beams 500 mm wide with openings for improving the ventilation. The angles of the frames and the joints with the cross-piece are reliably strengthened by riveting. The two frames are connected together by vertical H-beams.

With the new framework, the dimensions of the lower structure of the furnace have been considerably widened. The cross-section of the regenerators has been increased from 16 to 19.35 sq. meters, and the volume of the checkers from 106.3 to 133.5 cubic meters, with a slight increase in height. The volume of each slag pocket has been increased from 34.5 to 71.3 cubic meters and is adequate for 550 to 600 heats. The walls of the regenerators and slag pockets are thermally insulated by foamed fireclay.

Fig. 2. Longitudinal section and cross-section of 110-ton open-hearth furnace after reconstruction.

The "Simplex" butterfly type of reversing valve with a cross section of 1.3 × 1.3 meters caused a considerable reduction in draft due to high resistance and inadequate seal. During reconstruction, a double sliding type of valve was fitted, giving a through cross section of 1.5 × 2 meters.

Increasing the fundamental dimensions of the furnace, widening a number of cross sections and providing the new valve has considerably increased the thermal capacity of the furnace. The calculated maximum thermal capacity has been increased from 20 to 24 million cal/hour. As the result of reconstruction, the average charge of the furnace has been increased by 23.5 tons (23%), while the duration of a heat has risen by only 2%, chiefly on account of increased charging and melting time (see table). The productivity of the furnace in a nominal 24 hours has increased by 20%, with simultaneous increase from 27.4 to 42% in the proportion of more difficult alloy steels melted. The life of the furnace has been materially increased, and the length of a campaign has been increased by 20%.

BLOWING OUT DUST FROM CHECKER FLUES

Engineer V.N. Kazantsev

Shift Heat Engineer at Open-Hearth Steel Plant No. 1 of the Kuznetsk Metallurgical Combine

After conversion of the open-hearth furnaces to checkers made of basic refractories, the service life of the upper furnace structure was increased and exceeded the service life of the furnace bottom (slag pockets and
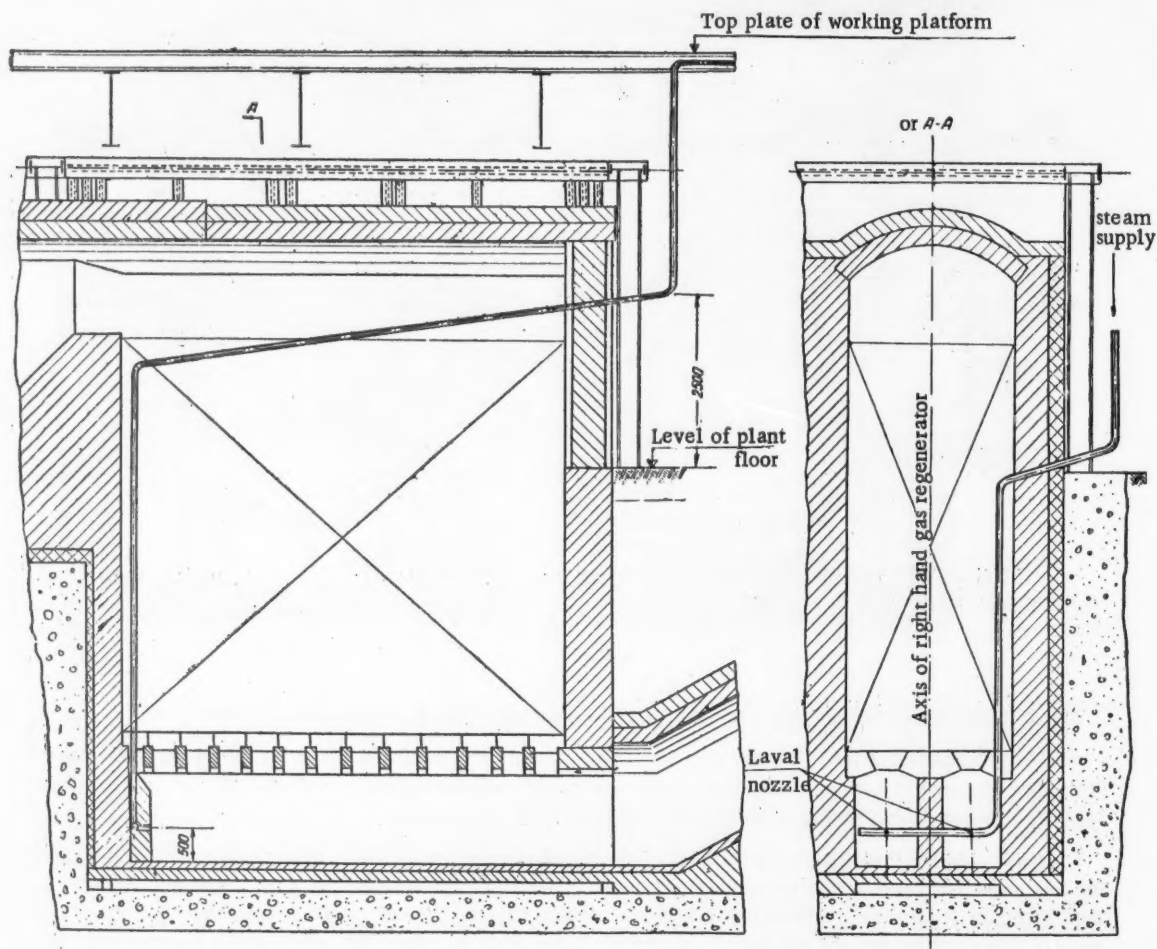

Fig. 1. General view showing steam supply for blowing dust out of the checker flues of an open-hearth furnace.

regenerators). While the problem of removing slag from the slag pockets during the working of the furnace has been partly solved in our plant, the problem of cleaning the regenerators from furnace dust has not yet been solved.

The open-hearth furnaces at out plant, with the exception of one, are heated by a mixture of coke-oven and producer gas. The calorific value of the coke-oven gas is 4200–4300 cal/normal cubic meter, and of the producer gas 1450–1550 cal/normal cubic meter.

The producer gas enters the furnace uncleaned at a temperature of 450–500°C. Its consumption is not measured, but according to calculation, the average value for this per furnace is about 7000 normal cubic meters/hour. The producer gas contains a considerable amount of carbon dust (soot), which does not all settle in the dust catchers but enters the checker flues with the gas. These flues rapidly become blocked not only with furnace dust but with soot. In particular, much soot settles in the checker flues of the gas generators of furnaces situated close to the gas-producer plant (Nos. 2, 3 and 4 furnaces), where the producer gas has the shortest distance to travel.

Years of experience in operating furnaces on producer gas shows that the checker flues are often blocked to the top with soot and dust. This results in a serious drop in the thermal efficiency of the furnace towards the end of the campaign.

The removal of dust from the checker flues is of great importance, not only with regard to improved working of the furnace, but also to reducing the labor of workmen during the repair of furnaces.

In the working process of the furnace, the dust suspended in the producer gas settles in the checker flues because of the change in direction of the gas. In order to remove the dust, the latter must be brought into suspension again. This can only be done by blowing a powerful jet of air or steam along the flues. The direction of the jet must be that of the movement of the discharging combustion products. Our plant experiences a considerable shortage of compressed air so that this cannot be used for dust removal. Superheated steam at a pressure of 10–12 atm. and a temperature of 200°C was used for blowing out the checker flue dust.
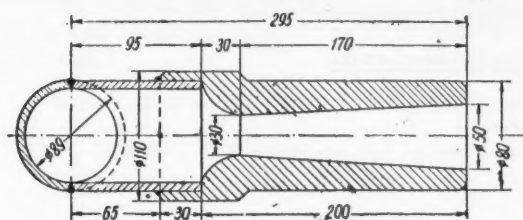

Fig. 2. Connection of Laval nozzle to steam pipe.

An experimental installation for blowing out the checker flues with steam was mounted on No. 5 furnace in February of this year during an intermediate cold repair (Fig. 1). The steam is supplied by a 3" stainless steel pipe laid in the checkers along the wall. At a distance of 500 mm from the bottom of the flues, the pipe is bent at a right angle to form a cylindrical manifold closed at one end. Welded to this manifold, along the axes of the flues, are two Laval nozzles (Fig. 2), one in each flue. The steam jet is thus directed away from the wall toward the gas stack flue. The calculated performance of the Laval nozzle is steam consumption 200 kg/hour and steam outlet velocity 700 meters/sec.
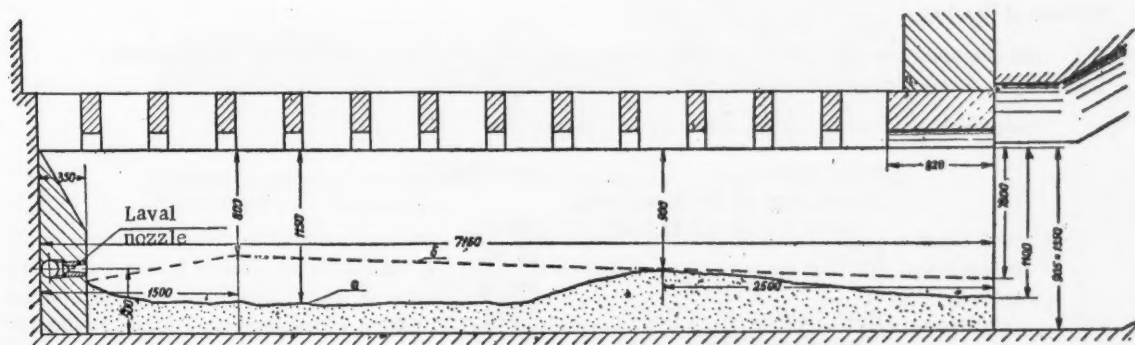

Fig. 3. Distribution of dust in checker flue: a) right-hand steam blown gas regenerator; b) left-hand gas regenerator.

No. 5 furnace was laid off for hot repairs at the 247th heat. During the intervening period, the dust had been blown out four times: at the 85th, 186th, 242nd and 243rd heats, during the melting and refining periods and only from the checker flues of the right-hand gas regenerator. The flues of the left-hand regenerator were not blown out due to failure of the steam supply during the first blowing operation at the 85th heat.

This enabled the effectiveness of blowing out the checker flue dust to be evaluated. The average length of time taken in blowing a flue with steam was 12 minutes.

After the furnace had been laid off for hot repairs and the checker flues had been inspected, the volume of dust deposited in them was measured (Fig. 3). It was found that in a flue of the right-hand regenerator, the volume of dust remaining was 37% less than in the flue of the left-hand regenerator. In the flues of the right hand gas regenerator, the accumulation of dust reached a height of 450 mm at a distance of 2.5 meters from the outlet to the stack flue. This shows that one nozzle per flue is not sufficient for an effective removal of the accumulated dust. A second additional nozzle must be mounted at a distance of 2.5 meters from the outlet of the checker flue to the stack flue with a separate steam supply.

The experiment on the removal of dust from checker flues shows that:

(1) steam blowing of the checker flues gives a positive effect;

(2) to increase the effectiveness of blowing out the checker flue dust, two Laval nozzles per flue must be provided;

(3) for reliable operation, the stainless steel pipe system must be laid along the reinforced concrete casing of the regenerator, i.e., the steam pipeline must be outside the high temperature zone;

(4) dust can be blown out of the air regenerator flues by means of steam.

ALUMINA-CHROME REFRACTORIES FOR LINING STEEL PLANT LADLES

Cand. Tech. Sci. V.A. Bron and Engineer P.A. Lande

The firebrick linings of steel plant ladles at the Chelyabinsk iron and steel plant do not last more than six to ten heats. In order to improve ladle life, trials were, therefore, carried out with alumina-chrome brick produced at the Semiluksk refractories plant and characterized by a high Al_2O_3 content and good thermal stability. The refractoriness of these bricks and the initial softening temperature under load are markedly superior to those of fireclay.

The chemical composition of alumina-chrome brick is: 77.38% Al_2O_3, 6.70% SiO_2, 10.81% Cr_2O_3, 1.17% Fe_2O_3, 1.02% MgO, 3.29% R_2O.

The brick has the following physico-chemical properties:

Refractoriness	over 1800°C
Temperature of initial softening under a load of 2 kg/cm^2	1560°C
Porosity	19.6%
Compression strength	824 kg/cm^2
After-contraction at 1500°C	0.32%
Thermal shock index, water quenching	over 35
Thermal conductivity	1.79 cal /m/hr/°C

The bricks were tested in 100 ton ladles.

In the preliminary trials, the alumina-chrome bricks were used in small patches in the fireclay lining in several ladles.

Observations of the lining of one of the ladles after service showed that, after four heats, the patches of trial brick had a markedly greater residual thickness than the fireclay lining. The trial patches clearly stood out from the fireclay surface. In fact, they projected 30 to 40 mm while, after six heats, the alumina-chrome brick stood out 50 to 60 mm from the fireclay lining. After eight heats, the ladle had to be repaired because the fireclay lining was almost completely worn down.

The superior wear resistance of the alumina-chrome brick was also apparent in the remaining trial ladles. A ladle was tried with the bottom and lower 13 rings of the side wall bricked completely in alumina-chrome and the remaining lining in firebrick.

Because of unsatisfactory drying and heating up of the ladle, vertical cracks occurred in the alumina-chrome part. In spite of this, the alumina-chrome lining was completely satisfactory as regards stability. After seven heats, the firebrick portion of the lining was completely worn out (residual brick thickness in the middle rows was 5 to 15 mm). Wear of the firebrick lining was as much as 100 mm or 12 to 15 mm per heat. The residual thickness of the alumina-chrome brick, however, which was in the most severely worn lower rows of the lining was not less than 110 to 115 mm, i.e., wear did not exceed 1 to 2 mm per heat. When the firebrick lining was removed, the alumina-chrome bricks which were strongly adhering to it, were chipped off.

Overall lining life of the ladle walls was 7 heats. The ladle bottom bricked with alumina-chrome was left for a second campaign. After two campaigns (14 heats), the residual brick thickness of the bottom was 110 to 135 mm.

Some difficulties associated with alumina-chrome refractories were encountered during ladle operations:

1) slagging of the brick surface and partial spalling of the brick;

2) difficulty in removing slag because of its high viscosity;

3) partial skulling of the ladle.

Residual thickness of the fireclay; 1) alumina-chrome; 2) bricks from one and the same area of lined ladle.

A trial was made of a ladle completely lined with alumina-chrome brick. To improve thermal insulation, two layers of asbestos were installed to a height of 13 to 14 rows between the ladle jacket and the outer (nonoperational) ladle lining. In order to maintain the capacity of the ladle, the KP-12 brick (working thickness 150 mm) was replaced by the KP-11 (working thickness 115 mm). Drying and heating up were carried out more carefully to avoid the formation of cracks in the lining. The lining lasted 14 heats without repair to ladle walls below the tenth row from the top. The top ten rings were replaced by firebrick after the ninth heat to facilitate slag removal.

After fourteen heats, the ladle was taken off for repairs because of a great amount of skull in spite of the fact that the ladle walls remained thick and strong. Residual thickness of the lining at the top of the ladle was 87 to 120 mm, in the lower rings, it was 80 to 100 mm and in the "combat area", it was 38 to 60 mm. In similar places such as the "striker patch" where the metal stream impinged on the lining, the firebrick was only 10 mm thick, even after six to seven heats.

It may be assumed that the lining of the trial ladle could have withstood an additional six to eight heats, i.e., the total life would have been 20 to 22 heats. The average monthly life of 100-ton ladles with firebrick linings during the trial period was 7.6 heats.

The following factors had an adverse effect on alumina-chrome linings:

1) The lesser thickness of the protective layer of ladle lining consisting of a single course of "bedding" 40 mm thick.

2) The dimensions of the KP-11 and KP-12 brick (to Russian standard GOST 5341-50) are unsuitable for the profile of 100-ton ladles as a result of which the lining joints are too thick.

3) The partial skulling of the ladle lined with alumina-chrome refractory is caused by the high thermal conductivity of the trial brick and inadequate thermal insulation of the ladle. Thermal insulation of the ladle can be improved by increasing the protective layer and making part of it of light weight brick while, at the same time, reducing the thickness of the working part of the lining to 80 to 100 mm.

The trials led to the following conclusions:

1) Alumina-chrome brick used in steel plant ladles exhibited high wear resistance, greatly exceeding that of firebrick. Ladle lining life is at least doubled.

2) It is essential to carry out more research with alumina-chrome brick for steel plant ladles and to clarify the optimum conditions for its use. In particular, efforts should be made to try to reduce the thickness of the working lining down to 80 to 100 mm and, on the basis of this reduction in thickness, to increase the thickness of the protective lining of light-weight firebrick so as to improve thermal insulation of the lining.

3) Manufacturing techniques for alumina-chrome brick for steel plant ladles must be perfected in order to reduce the thermal conductivity of the refractory.

STEEL LADLE OF LIGHT CONSTRUCTION

Engineers M.E. Krivitsky and G.A. Dubrovin

"Zaporozhstal" Works

It is well known that in many open-hearth steel plants having 185-ton furnaces and above, standard steel ladles are employed, having a capacity of 180 and 200 tons of metal tare factor (ratio of weight of ladle without lining to total weight of molten steel) of 24-25%. These ladles were designed as long ago as in the 1930's for typical teeming cranes having a weight-lifting capacity of 220 and 260 tons. Up to the present time, there has been no modification in the construction of these ladles.

At the "Zaporozhstal" Works, the increase in the steel output since 1952 has been limited by the capacity of the steel pouring equipment. The problem had to be solved by increasing the capacity of the ladles. To increase the shell of the ladle for the purpose of increasing its capacity increases the weight of the filled ladle beyond the load-lifting capacity of the crane. A group of engineers at the works* proposed a new design for a lightweight steel ladle with a capacity of 220 tons and a tare factor of 15.8%. The steel part of the ladle was reduced in weight not merely by reducing the cross section of the ladle elements, but by a correct distribution of the stresses in them.

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*G.A. Dubrovin, M.E. Krivitsky, K.P. Gulyanitsky, N.I. Borodimov; the designing and mathematical part of the work was done by G.A. Dubrovin.
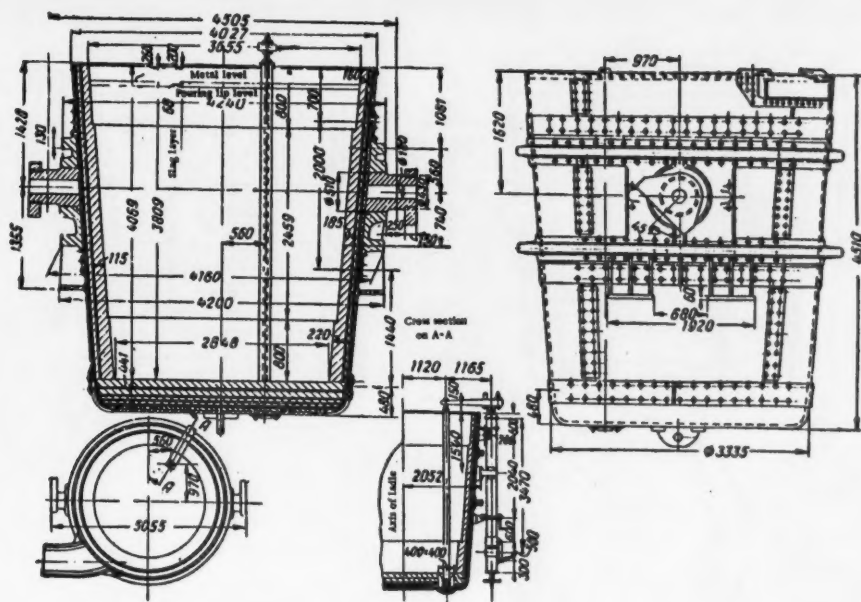
Fig. 1. General view of lightweight steel ladle (tare factor 15.8%).

Fig. 1 shows a general view of the first modification of ladle. The latter consists substantially of three butt-welded steel cylinders or rings. From the outside, the joints are covered by plates riveted to the cylinders. The ordinary riveted joint has its rivets at a much greater pitch than normal. The rivets are countersunk on the inside. The ladle bottom is flat with a flange of radius R = 162 mm, strengthened by reinforced plate. The bottom is butt-welded to the lowermost cylinder or ring of the shell, and this seam is also covered with plate, additionally reinforced by one row of rivets countersunk on both sides.

The principal structural modification of the ladle consists of special trunnion blocks cast integral with the stiffening rings. The trunnions are set in the blocks with reliable tightness.

The ladles have been subjected to cold and hot test with 40% overload on the trunnions. The load was applied to the trunnions by means of hydraulic jacks having a load-lifting capacity of 250 tons, with simultaneous internal hydraulic pressure in the ladle of up to 3.5 atm. The results of the tests showed that the stresses in the basic constructional elements were less than or equal to the calculated stresses and did not exceed 680 kg/cm^2.

The 220-ton ladles of the new light construction have been used in the steel plant since 1956. Their actual capacity when freshly lined is 219—220 tons, and after pouring 7 or 8 melts it becomes 224—225 tons. The ladles are lined by the accepted method used at the works.

At the present time, ladle construction has been modernized and a new all-welded ladle is being made (Fig. 2) with a convex bottom and a greater flange radius. The construction of the trunnion blocks and stiffening rings has been improved, and the latter are cast separately from the blocks. These modifications have enabled the capacity of the ladle to be increased to 230 tons with a tare factor of 11.4%.

The first two ladles of the new type have withstood the cold test of the works with 40% overload and simultaneous internal hydraulic pressure of up to 3.5 atm. In these tests, the actual (measured) stresses in the principal elements did not exceed 440 kg/cm^2. The two ladles have also withstood hot tests when filled with molten metal. The results of the tests have shown that the strength of the ladles is satisfactory. Six ladles of the new construction are currently in service and others are being made.

The increase in capacity of the ladles with simultaneous reduction in their weight has enabled the production of steel to be increased by about 12—13% without increasing the crane equipment.

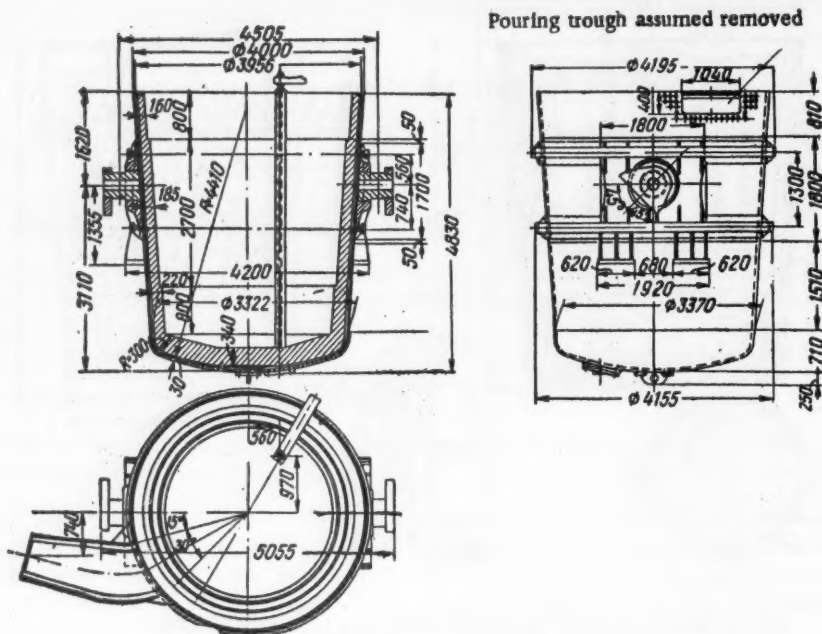Pouring trough assumed removed

Fig. 2. General view of all-welded steel pouring ladle (tare factor 11.4%).

The valuable experiment of the "Zaporozhstal" Works should be utilized by all iron and steel works.

TRAVELLING PLATFORM FOR STEEL CASTING PIT

In the open-hearth steel plant of the Revdinsk hardware metallurgical works, a travelling platform has been constructed for stripping ingots in the casting pit. The platform (sketch) consists of a steel body (1) supported by four wheels (2) on rails (3), on which travels the electric car with the steel ladle. The floor (4) of the platform is covered with corrugated steel. Welded to the lower part of the body is a polished metal sheet (5), the mirror surface of which reflects the greater part of the heat radiated from the hot ingots and molds. Such a sheet (chromium plated, polished or aluminum) is able to reflect 70 - 80% of the heat rays. The travelling platform has two protective barriers (6) and (7), the front one being 650 mm high and the back one 1500 mm. The barriers are made of angle irons and two metal sheets of 2 or 3 mm gage with an asbestos insert between them 10 mm thick. The weight of the platform is 950 kg.

Four men work on the platform. They hook the ingot molds on bars suspended from the hook of the overhead crane. When necessary, the platform is moved by hand by one workman.
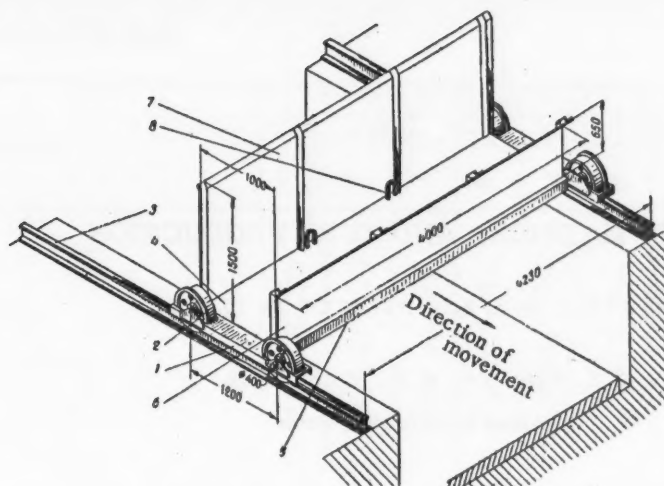
Travelling platform for steel casting pit: 1) body; 2) wheels; 3) rails;
4) floor; 5) polished sheet metal reflector; 6) front barrier; 7) back
barrier; 8) eyes.

After stripping the ingots, the platform is lifted off the pit by the crane and is put down in a vacant space at the end of the casting bay.

The use of travelling platforms for stripping the casting pit considerably lightens the labor of the work-men.

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ROLLED MATERIAL AND TUBE PRODUCTION

COMBATING SURFACE DEFECTS IN PLATES

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Enakievsk Metallurgical Works

One of the chief requirements which steel plates have to satisfy is high surface quality. This requirement is difficult to meet if plates are rolled from ingots having various surface defects, such as scabs, cavities, slag inclusions, blisters, etc.

At our works, plates of a thickness of 4, 5, 6 and 8 mm and a width of 1100–1400 mm are rolled from ingots weighing 250–650 kg on a Lauth three-high medium plate mill with rolls of diameters of 700/600/700 mm and a body length of 1800 mm.

The principal defects of the plate ingots cast at the works are scabs and cavities. Thus, out of 3060 ingots examined, 218 had scabs and 453 had cavities. In many cases, these flaws are retained in the surface of the plates, reducing their quality. It became necessary to establish the connection between the external defects of the ingots and the surface defects of the plates.

The investigation was made by the works employees. For this purpose, ingots with characteristic defects were selected and rolled. From the plates in which the ingot defects persisted, specimens were taken for investigation of the microscopic and macroscopic structure and the nature of the non-metallic inclusions.

Scabs

On the 218 ingots selected, the scabs varied in thickness and were situated mainly in the bottom part of the ingot (about 70% of the scabs).

Scabs of a thickness of 2.5–3.0 mm and above were regarded as coarse, and those of a thickness of less than 2.5 mm as fine. Coarse scabs were 30–40 mm across, while fine scabs were usually of small dimensions, and were situated on the faces of the ingot in separate scales or plates.

56 ingots with coarse scabs and 36 with fine scabs were selected for observation. In the rolling process, a large number of scabs of different sizes and shapes were observed on the plates.

Table 1 shows that coarse scabs persist on the plates mainly as scabs. During rolling, about 20% of them are converted into some other form of defect (cavities, grit), resulting from the detachment of the scabs from the surface of the plates. If there were non-metallic inclusions beneath the scabs on the ingot, grit was left on the plates instead of the scabs. If, however, the scab had a clean surface and during the rolling process became detached from the plate, it left a depression or cavity. Fine scab (scaly) in most cases is not retained on the plates.

In all cases, the scabs were accompanied by a crack filled with non-metallic inclusions of a light gray color. There was always a decarburized zone along the edges of the cracks.

Grit from under scabs appeared as yellowish-red spots on the surface of the plates, elongated in the rolling direction. The non-metallic inclusions of gray color were arranged in chains, showing that this defect was formed during the casting of the metal.

TABLE 1

Results of Inspection of the Plates

Ingot defects	Total No. of ingots	Defects persisting on the plates		Defects not persisting on the plates		Transformed into other defects					
						cavity		scab		grit	
		No.	%	No.	%	No.	%	No.	%	No.	%
Scab:											
course	56	37	66.0	7	12.5	7	12.5	—	—	5	9.0
fine	36	6	16.7	20	55.6	—	—	—	—	10	27.7
Cavity, depth mm											
3—5	36	—	—	30	83.2	—	—	4	11.6	2	5.2
6—9	34	1	2.9	26	76.2	—	—	7	20.9	—	—
over 9	30	—	—	17	56.6	—	—	13	43.4	—	—
Slag	33	6	18.2	13	39.4	—	—	8	24.2	6	18.2
Grit	22	22	100.0	—	—	—	—	—	—	—	—

Cavities

Cavities are extended depressions distributed over the wide faces of the ingot. Their dimensions usually vary: there are cavities of a length of 20—60 mm and a depth of 15 mm, but most often the depth is 5—8 mm. Out of 100 investigated ingots affected by cavities, 92 had cavities of a depth of up to 9 mm.

In most cases, the cavities have a sloping edge and are situated in the upper third part of the ingot (of 100 ingots, 69% with cavities in the top third, 23% in the middle and 8% in the bottom third).

Cavities occurring on ingots do not persist after rolling. About 10% of plates from ingots with cavities have a scab where the cavity was, and less frequently grit. A scab is formed by the rolling of the metal in the first few passes if the ingots have cavities with steep edges. If the cavity is not deep and the edge is sloping, it is rolled out during the rolling process and the surface of the plates obtained is clean.

Cavities of steel melting origin are usually shallow, with a smooth bottom and oval shape, elongated in the rolling direction. Sometimes slip lines can be seen on the bottom of the cavities. Non-metallic inclusions are situated in a compact, light gray strip or band; their microstructure is fine-grained.

Cavities of rolling origin are formed by pieces of steel from the top part of ingots falling on the surface being rolled. At the end of rolling, the pieces of steel spring off, leaving depressions on the plates. Cavities of rolling origin are round with an uneven bottom.

Grit

Ingots affected visibly by grit (pieces of brick from bottom pouring equipment and other non-metallic inclusions), are relatively few in number. The grit is situated mainly in the bottom part of the ingots. More often, such grit remains suspended inside the ingot or lies below other external defects (scabs, slag, etc.). As the thickness of the rolled product diminishes, the internal grit appears on the surface of the plates. Grit is therefore encountered more frequently on the plates than on the ingots.

Grit of rolling origin is due to pieces of bottom firebrick, which have adhered to the ingot, being rolled in. Externally, it does not differ from grit of steel melting origin and appears in the form of yellowish-red spots or patches, elongated in the rolling direction. The microstructure of steel in the vicinity of grit of open-hearth origin is considerably contaminated by non-metallic inclusions (oxides of iron and silicates), and grit of rolling origin appears as a deformed grain, caused by non-metallic substance of greater hardness being pressed into the plate.

Slag Inclusions

Slag inclusions on the surface of ingots appear in different shapes and sizes. Most frequently, slag on the faces occurs as small isolated inclusions, mainly in the middle of the ingot (51.5%). After such ingots have been

rolled, defects remain on the surface of the plates not only in the form of slag, but also as grit, if the latter was present beneath the slag inclusion, and as cavities formed by the rolling operation.

If the slag or oxide inclusions are broken out during rolling, depressions of a round or oval shape, "pitting", are left, this usually occurring on the sheets in isolated collections. Pitting is also caused by the opening up of gas blisters situated close to the surface. They open up during reheating, are filled with slag and, during rolling, with scale.

Microscopic examination of polished surfaces obtained from pitted plates showed that there is no increased contamination of the metal by non-metallic inclusions in the vicinity of the defect. The grains of the metal are deformed slightly, which may merely be due to the less plastic foreign bodies being forced into the plate during rolling.

Influence of Plate Thickness on Quality

The number of defects on the surface of plates depends to a considerable extent upon the degree of reduction of the rolled plate. Defective ingots were rolled to plates of a thickness of 4, 5 and 6 mm. Investigation showed that the smaller the thickness of the rolled plates, the fewer the number of scabs remaining on the surface, while the number of cavities formed increased. Thus, on 75% of the 6 mm plates, rolled from ingots affected by coarse scabbing, the latter was retained on the surface, while 25% of the plates had a clean surface. On 60% of the 4 mm plates, scabbing was retained, while 20% of the plates had cavities on the surface.

This regularity obtaining in the results can be fully explained. As the thickness of the rolled product is reduced, more and more scabs spring off the surface of the plates, leaving cavities behind them. The subsequent behavior of the cavities depends upon their size and shape, and also upon whether they have been formed at the commencement or end of the rolling process.

When rolling the 4 mm plates, grit affected 12.4% of the plates examined, the corresponding figures for the 5 and 6 mm plates being 10.7 and 3.4%, respectively.

Influence of Method of Heating the Ingots

The soaking pit of the medium plate mill, before reconstruction, operated with heating from one end, the ingots were not turned over in the furnace and therefore the side of the ingot directed towards the roof (top) was subjected to greater oxidation during heating than the bottom. A thicker layer (3−4 mm) of scale is formed on the upper side than on the lower side, where the scale has an average thickness of not more than 1.5 mm. In most cases, the external defects on the upper side are burnt out.

TABLE 2

Influence of the Method of Heating the Ingots

Position of defects on plate	2nd quality according to metal	Defects on plates			
		Scab	Cavity	Grit	Slag
Top side:					
Number	46	12	7	20	5
%	2.1	0.6	0.3	0.9	0.2
Bottom side:					
Number	147	67	3	40	6
%	6.8	3.1	0.1	1.9	0.3

To examine this question, some of the ingots with different external defects were placed in the furnace with the affected side up, and the others with that side down. The results of an inspection of 2162 plates are shown in Table 2, from which it follows that a larger number of defects are retained on the plates if the ingots are set in the furnace with the defective side on the bottom.

On the upper side of the plates, there was only 0.6% of scab, but on the lower side there was 3.1%.

To ascertain what kinds of external flaws on the plates are defects of rolling origin, 57 clean ingots were rolled: 20 of them had been placed with an absolutely clean face on the soaking pit hearth and 37 with that face directed towards the roof. Careful inspection of the plates rolled from these ingots failed to reveal any defects, with the exception of five plates which had to be dressed on account of grit but which were accepted as of first quality. Thus, when ingots which have no surface defects are rolled, the resulting plates have a clean surface.

For the purpose of examining ingots which had had their external defects removed by dressing, 20 ingots with main defects—scabs and cavities—were selected. All the defects were measured. While the defects were being removed, the character and quality of dressing was observed. Of the 20, only one ingot had an incompletely removed scab. All the dressed places on the ingots had sloping edges.

Examination of the plates rolled from these ingots showed that only the ingots with incompletely dressed scabs produced second-quality plates, judged by scab and grit. The other plates were of first quality. The surface of plates can consequently be considerably improved by high-quality dressing of the defects on the ingots.

INCREASING THE PRODUCTIVITY OF STRIP SLITTING DISK SHEARS 420

A.P. Koshka

Novosibirsk Metallurgical Works

Disk shears 420 in the middle bay of the cold-rolling shop of the Novosibirsk metallurgical works failed to ensure completion of the rolling shop plan, due to inadequate mechanization. In addition, it was necessary to eliminate another bottleneck in the shop, the pickling lines, where metal not requiring to be pickled was also slit.

Disk shears 420 intended for edge trimming and for slitting coiled strip having a width of 700 mm and thickness 1.5—4 mm into 2—15 strips, consisted, prior to mechanization, of a box type uncoiler, disk shears with edge trimmer, press table and a drum type recoiler (Fig. 1).
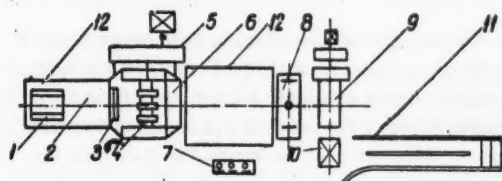

Fig. 1. Diagram of disk shear 420 unit before mechanization: 1) box type uncoiler; 2) manually operated shears; 3) tensioning rollers; 4) disk shears; 5) multiple stage reducing gear; 6) edge trimmer; 7) principal control panel; 8) press table; 9) recoiler; 10) lifting table; 11) slide; 12) flooring.

With such a construction of the unit, the complete cycle of slitting the steel can be divided into nine operations (Fig. 2).

An operational graph based on exact timing and photography of the working operations showed that the machine time (slitting) amounts altogether to 2 min. 15 sec. or 11.3% of the total time spent in slitting one reel.

Much time was spent on preparatory work. Thus putting the coil in the uncoiler by overhead crane took 2 min. 10 sec. Due to heavy loading of the cranes, the disk shears were inoperative for up to 15% of the total time, because of failure to supply the unit with metal. Before the strip was fed to the disk shears, the front end was cut off by hand with lever shears, which took 1 min.
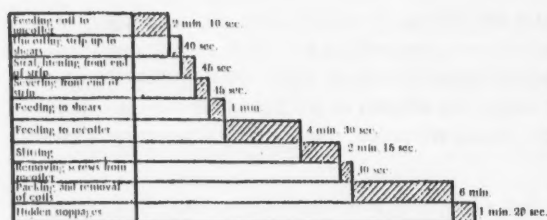
13 sec. Uncoiling was also done by hand as far as the tension rollers. Slitting of a following coil was commenced only after complete removal of the coil from the cradle of the recoiler hoist and when the latter had returned to its starting position.

Feeding coil to uncoiler		2 min. 10 sec.
Starting strip up to shears		40 sec.
Straightening front end of strip		45 sec.
Severing front end of strip		15 sec.
Feeding to shears		1 min.
Feeding to recoiler		4 min. 30 sec.
Slitting		2 min. 15 sec.
Removing screws from recoiler		30 sec.
Packing and removal of coils		6 min.
Hidden stoppages		1 min. 20 sec.

Fig. 2. Graph of operations for slitting a coil of un-
pickled strip 1.75 × 600 ⟶ 1.75 × 40 mm be-
fore mechanization.

It thus appeared possible that a considerable mar-
gin of increase in productivity of the disk shears and
easing of the labor conditions could be secured by eli-
minating the bottlenecks and mechanizing the heavy
labor. With this object in view, the slitting unit was
provided with a number of additional contrivances and
mechanisms (Fig. 3), as a result of which the time re-
quired for the individual operations was cut down con-
siderably (Fig. 4).

The entry end of the unit was equipped with an
inclined roller coil collector, accommodating four coils
and ensuring rhythmic operation of the unit. The col-
lector consists of two roller sections mounted on a rigid
frame and equipped with a brake device operated by a compressed air cylinder situated under the sections.

The provision of the collector has resulted in the complete elimination of stoppages of the unit due to
failure in supply of strip. The cranes have been relieved to a considerable extent, since it has been found pos-
sible to set down two coils at one lift.
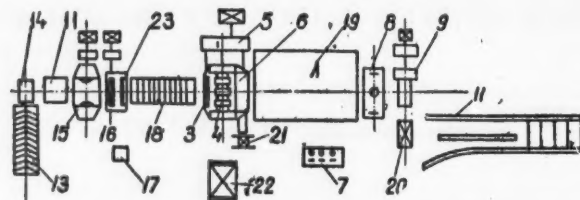
Fig. 3. Diagram of disk shears unit 420 after reconstruction:
13) coil collector; 14) coil overturning device; 15) conical
uncoiler; 16) feed rollers; 17) auxiliary control panel; 18) roll
table; 19) auxiliary control post; 20) carriage; 21) skids;
22) box; 23) hydraulic shears (remainder as for Fig. 1).

At the exit end of the collector is a coil overturning device of the tilting type for receiving the next coil
from the collector and feeding it to the uncoiler. All this has enabled the time required for feeding a coil to the
uncoiler to be cut to 1 min. 15 sec.

Feeding the coil to uncoiler		1 min. 15 sec.
Feeding to feed rollers		57 sec.
Cutting the end		45 sec.
Feeding to shears		43 sec.
Feeding to recoiler		4 min. 25 sec.
Cutting		2 min.
Coiling rear end		1 min.
Removal from recoiler		30 sec.
Stoppages		30 sec.

Fig. 4. Graph of operations for slitting one coil of un-
pickled strip 1.75 × 600 ⟶ 1.75 × 40 mm after
mechanization.

The box type of uncoiler has been replaced by a
reel type of uncoiler, consisting of two movable man-
drels mounted on a stand and a hoist, for lifting the
coil to the level of the cones of the uncoiler mandrels
through an inner opening. In one of the mandrels, the
cone is driven.

Following the uncoiler is a roll feeder (suggestion
of Comrades Brusilovsky, Ravilov and Nazarov) for
gripping and feeding the strip to the hydraulic shears
and then, along the roll table, to the disk shears. The
roll feeder is a cast iron stand with guides, along which
moves a support with a roller. The upper, driven roller
is fixed to the upper part of the stand. As soon as the
end of the strip enters the gap between the rollers, the
lower support with its roller begins to move upward
under the action of a compressed air cylinder and pinches the end of the strip between the rollers.

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On the shafts of each of the rollers are mounted pinions having 24 teeth. When the lower roller occupies the upper position, the pinions are in mesh and the rotation of the upper driven roller is transmitted to the bottom roller. The rollers revolve in different directions but at the same speed.

Fixed to the feeder stand are the hydraulic shears (suggestion of Comrade Mordinov) for cutting the front and rear ends of the strip and also for cutting out weld seams.

The shears consist of a welded stand with guides along which moves a lower support with curved blade. The lower support is driven off a hydraulic cylinder connected to the existing hydraulic system of the recoiler hoist.

After the front end has been severed, the strip is fed to the disk shears by the feeder along a roller table mounted between the hydraulic shears and the disk shears.

TABLE 1

Duration of Work of Crews in Slitting one Coil of Hoop Iron

Operations	Time required to complete each operation, min. – sec.			
	Leonov	Kandri-mailo	Skakalsky	Progressive working group
Feeding coil to uncoiler . . .	1—13	1—13	1—20	1—13
Feeding strip to feed rollers .	0—48	1—15	0—48	0—48
Severing end of strip	0—45	0—45	0—45	0—45
Feeding strip to shears	0—50	0—37	0—42	0—37
Feeding to recoiler	4—29	3—58	4—50	3—58
Slitting	2—17	1—40	1—40	1—40
Coiling rear end.	1—10	0—40	1—09	0—40
Removal of metal from recoil.	0—25	0—35	0—31	0—25
Packing	2—30	3—24	3—30	2—30
Complete cycle	14—27	14—07	15—15	12—36

Mechanization of this section of the unit has made it possible to eliminate completely the heavy work and to reduce the time required to sever the end of the strip to 45 sec. and the feed time to 1 min. 40 sec.

A truck has been provided at the recoiler for moving the coils of slit metal from the recoiler drum. The truck with the metal is taken to the packing place and the hoist cradle is returned to its original position. The time taken for packing and removing the metal overlaps the period for slitting and other operations. The use of a truck has resulted in a saving of time of 6 min.

In December 1956, at shears 420, the removal of clippings from the edge trimming shears was mechanized, which has helped considerably in increasing the hourly production for the first six months of 1957 (by 233.3% compared with 1950).

Mechanization of the removal of clippings is as follows. Under the edge trimmer is mounted an inclined slide or chute with a throw-off end, along which the clippings pass into a steel box 1500 x 2000 x 1500 mm. Previously, the clippings were collected by hand; first on a carrier and then put into a box set up near the shears. At present, the pit is covered over, the cover serving as working platform.

After mechanization of the disk shears had been carried out for increasing their production, an investigation was made of progressive methods of working with a view to their adoption. Exact time measurements were made of a complete cycle of operations in slitting a coil carried out by three crews producing identical sections.

TABLE 2

Time Required by the Various Crews for Adjusting Disk Shears 420, min. — sec.

	Crews			
	Khandri-mailo	Skakalsky	Leonov	Progressive graph after adoption
Removing the guides:				
Upper	—	2—50	1—00	—
Lower	1—30	—	3—10	1—30
Unscrewing the fixing nuts .	2—50	3—20	4—00	2—50
Removal of front mandrel. .	3—10	3—10	3—10	1—00
Removal of assembly of disk shears	11—15	12—00	13—00	11—15
Wiping spindles	1—00	1—00	1—10	1—00
Preparing assembly	5—00	6—30	6—00	—
Mounting the assembly . . .	42—00	42—00	43—00	25—00
Returning the mandrel	3—00	3—00	3—00	3—00
Tightening the nuts.	5—50	7—00	6—15	5—50
Checking the clearances	5—10	6—00	5—50	2—10
Mounting the guides:				
Upper	4—30	5—10	4—50	4—30
Lower	1—30	—	—	—
Total.	1 hr. 26 min 45 sec.	1 hr. 32 min	1 hr. 34 min 25 sec.	58 min. 5 sec.

From the comparative tables (Tables 1 and 2) compiled on the basis of the time measurements, it may be concluded that the different crews carry out the same operations in different ways, and the sequence of carrying out the operations is also different. The best results were obtained by the crew led by Comrade Khandrimailo.

A general graph of the work was prepared on the basis of the operations which took the least time when carried out by any of the crews. The best methods were studied by the crews and adopted for production.

In addition, the quickest method of adjusting the shears was perfected and adopted.

The increase in productivity of disk shears 420 has saved the works 42000 rubles.

The steps taken have enabled the hourly output of the shears to be doubled (the productivity for 1950 being taken as 100%):

Year	1951	1952	1953	1954	1955	1956	1957 1st 6 months
Productivity of shears, %	124.4	125.9	144.4	174.0	203.7	203.7	223.7

INCREASING THE DRAWING RATE OF A 7.5-TON DRAW BENCH

V. A. Bogatyrev and I.N. Isaev

Nikopolsk Southern Tube Works

A group of workers from the tube-drawing plant at our works has done some work on the possibility of increasing the drawing rates without regulating devices, i.e., for constant chain velocity. As result of this work, it was found that it is perfectly possible to draw tubes without a mandrel at a rate of 40−45 meters/min. Due to the high chain velocity, however, it became difficult to ensure reliable engagement of the plyer hook.

Following a suggestion made by the manager of the equipment plant, A.A. Zhuravlev, the pitch of the chain was increased from 75 to 100 mm, whereby normal engagement of the hook in the chain was possible without complicating the construction of the plyer. The technical council of the plant proposed to reconstruct ten draw benches of the cold drawing section and to employ increased drawing rates of 46− 50 meters/min. The reconstruction of a double chain 7.5 ton draw bench by the provision of chains and sprockets of increased pitch has currently been completed; the drawing rate is 45 meters/min. Work has been carried out on the adoption of the process of drawing tubes with a wall thickness of 0.85−1.0 mm at this speed using a short mandrel.

The drawing tool has produced serious difficulties in regard to the use of high drawing rates. A work analysis has revealed systematic losses of time and poor surface quality, due to low strength of the dies. Photo-chronometrical observations have shown that the increase in the losses amounts to from 5 to 12.5% of the working time. The life of the drawing dies has been considerably increased by making them of 40Kh5T steel.

The plant is currently commencing work on the use of more durable dies of hard carbides. This will reduce stoppages of equipment and considerably increase the surface quality of the tubes.

The existing standard, GOST 2330−49, for hard carbide dies does not permit tubes to be drawn with considerable deformation, due to shortcomings in the construction of the dies. By agreement with the management of the All-Union Office of Technical Assistance on the Use of Hard Carbides, however, a plan has been drawn up for the technical conditions governing the production of dies with increased height and modified configuration of the entry taper (according to a standard proposed by the Southern Tube Works).

An experimental set of pressing molds has already been made and in the near future the plant will be provided with a high quality tool, satisfying all requirements for changing over to increased drawing rates.

OPERATION OF MILL 600 ACCORDING TO A NEW SCHEDULE

Deputy Chief Engineer M.V. Shuralev

Engineer — Roll Designer S.G. Nekrasov

(Zlatoust Metallurgical Works)

On mill 600 in our Works, low-carbon and alloy steel circular sections of 90—150 mm diameter, and blooms of square cross section 170 by 170 and 190 by 190 mm and of 520—780 kg weight are rolled.

The mill consists of four three-high stands in tandem arrangement; the diameter of rolls is 600 mm and the length of roll barrel is 1600 mm. Billets are heated in two holding furnaces.

According to the former roll design, stand I was provided for the production of the starting square section (Fig. 1,a) for all the circular sections (from 90 to 150 mm diameter).

Stand II contained finishing circular passes of 90—115 mm and two pre-finishing flat oval passes: one for the circular sections of 90—95 mm diameter and the other for the circular sections of 100, 105, 110 and 115 mm diameter.
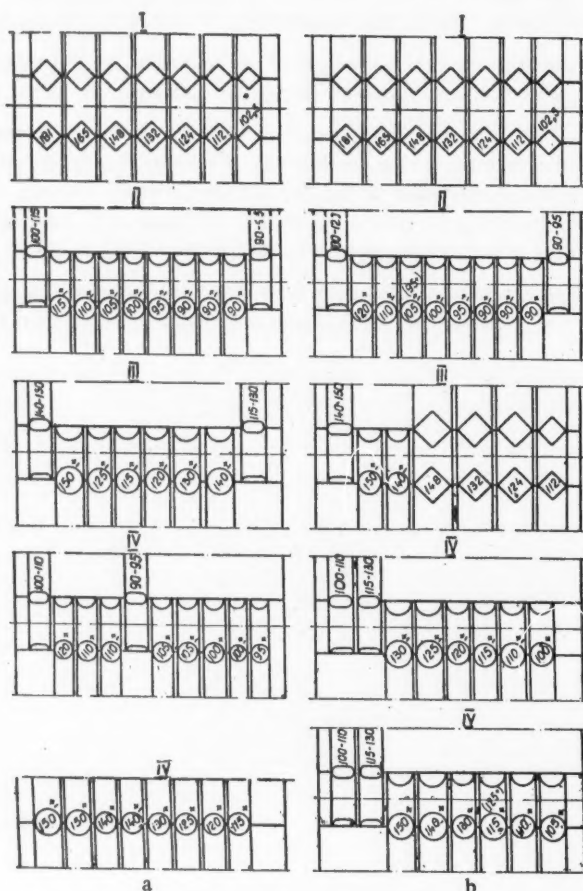
Fig. 1. Roll design of mill 600: a) old; b) new.
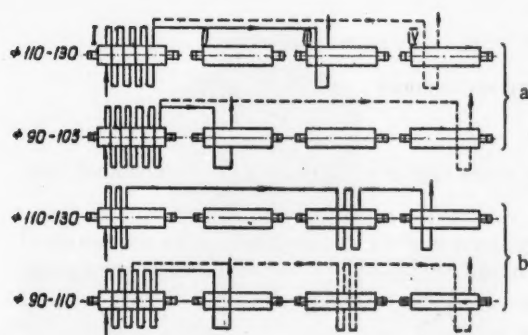
Fig. 2. Rolling scheme of circular section: a) old
b) new.

The finishing circular passes for the rolling of circular sections of 115—150 mm diameter were on the rolls of stand III.

The passes for the rolling of circular sections with positive tolerances were arranged in two sets of rolls in stand IV.

When the billets are heated in two furnaces, it is expedient to carry out rolling of two sections of different dimensions simultaneously. However, the rolling of billets of one steel batch and of the same size from two furnaces simultaneously, makes difficult the adjustment of the mill for one section because of the difference in metal heating. The difficulty in the adjustments becomes greater still when various grades of metal are rolled.

The rolling of two different sized sections was found to cause a very irregular distribution of passes among the stands. In the first stand the billets were rolled in 7 to 11 passes down to the required square which was then transferred to stands II, III or IV — depending on the diameter of the section rolled — for the final passes: oval and round (Fig. 2,a).

In such a scheme the first stand was the most overloaded, two other stands were little used and one was continually running idle. It was nearly impossible to work with overlapping.

Moreover, the fact that two rough square sections had to be obtained simultaneously from stand I caused complications in the adjustment of the mill and made the control of the dimensions of the final section difficult.

In order to increase the operating efficiency of mill 600 the rolling scheme was therefore changed. In the new scheme the passes of stands I and II were not substantially modified; four reduction passes, one oval and two finishing circular passes were introduced in stand III and circular passes in stand IV (Fig. 1,b).

The distribution of passes among the stands and the design of passes in stand III are arranged in such a manner that the 90—110 mm sections can be rolled on the stands I and II and the 110—130 mm sections as well as those of 100—150 mm with positive tolerances, on stands I, III and IV (Fig. 2,b).

When two sections from different groups (for example, circular sections 90 mm diameter and 115 mm diameter) are rolled simultaneously, the billets are heated in two separate furnaces.

Fig. 3. Schedules for rolling of circular sections:
a) old; b) new.

Billets for the 90—110 mm diameter section are rolled down to the required starting square in stand I and then transferred to the stand II where two finishing passes are made. When 110—130 mm sections are rolled, five passes are made in stand I and two to four passes in stand III until the required starting square section is obtained; finishing passes (oval and circular) are made on stand IV. The required starting square section for the 140± and 150± mm diameter sections is obtained on stand I and finishing passes are made on stand III.

Operation according to the new rolling scheme increased output of the mill owing to the more uniform loading of the stands and the working with overlapping (Fig. 3 compares the schedules of rolling of 115 mm diameter sections according to the old and new methods).

Modification of the rolling scheme facilitated the adjustment and the control of the mill because the required starting square sections are obtained in separate stands.

The change-over to the new rolling schedule has saved the works about 500,000 rubles annually.

INSTRUMENT FOR SLOT CUTTING IN THE NEW RAIL BACKINGS

R.P. Radchenko and V.N. Kovalskaya

(Kuznetsk Metallurgical Combine)

In April, 1956, our Combine began the production of a new type of railroad rail joint to reinforced concrete sleepers.

One of the most labor-consuming operations in the production of these joints proved to be the cutting of slots of a complex shape in backing plates. Moreover, the flange thickness had to be considerably greater than in the conventional backing for the R−50 rails. These factors made the cold slot cutting difficult and prevented the production of smooth walls in the slots. Hence, in order to reduce the stresses on slot cutting in 26 mm thick flanges as well as to obtain smooth walls without cracks, it became necessary to heat the material to 500−600°C.

The heating of the material, however, had a detrimental effect on the durability of the cutting instrument, in particular on the punches of a complex design (Fig. 1) made of steel U10. On punching hot metal, the cutting edges of the puncher became tempered. Because of reduced hardness the punches became distorted and were quickly rendered useless. The punches made of steel U10 cut only 80−100 slots while the operating efficiency of the punching machine was 1500 backing plates per shift.

The cutting edges of the punches were reconditioned by electrowelding with high-speed steel. As, however, a thermal treatment of the welded-on layer was not possible (the basic metal was not resistant to tempering) the reconditioned punches cut 300−600 slots. Such a durability of the instrument did not allow an increase in the efficiency of the punching machines and necessitated frequent mechanical treatment and machining of punches resulting in high production cost.

In order to increase the life of punches in their original state as well as after the reconditioning of the cutting edges, the Combine carried out development work regarding the choice of steel. After the testing of several grades of steel, 5KhNV steel was selected, as it showed the best results. The chemical composition of this steel is as follows, %:

C	Mn	Si	P	S	Cr	W	Ni
0.56	0.64	0.48	0.015	0.005	1.07	0.90	1.52

The punches made of steel 5KhNV, oil-quenched from 860°C and tempered at 240°C for 1.5 hours, cut 800−1000 slots and became unserviceable only as a result of natural wear of cutting edges of the working part (Fig. 2). No crumbling or crushing of the cutting was observed.

The cutting edges of the punch were reconditioned by electrowelding with high speed steel R18 in a strictly determined order. Prior to welding, the working part of the punch was heated to 250−300°C in order to prevent cracks in the welded-on layer.
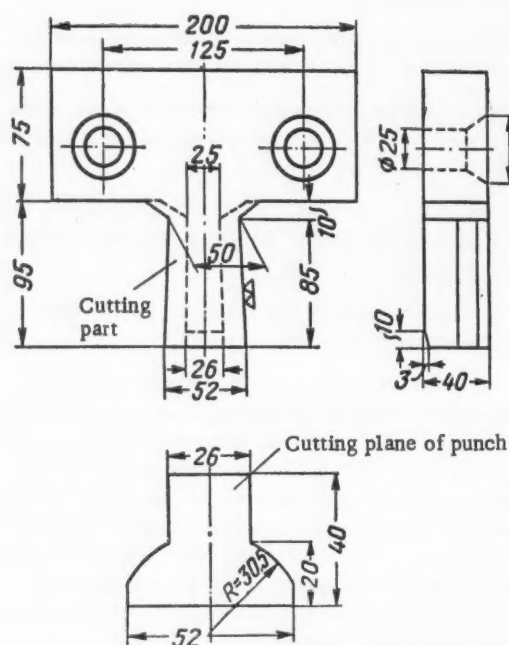
Fig. 1. Punch for slot cutting in backings for reinforced concrete sleepers.

One-layer welding by a continuous fillet weld was started from the large side at the distance of 20 mm from the end (Fig. 3, point 1), then the square part of the punch was welded on (point 2) followed by the corner (point 3). The end plane was welded on last. The welded part of the punch was tempered three times in a saltpeter bath at 560°C for one hour with subsequent air cooling.
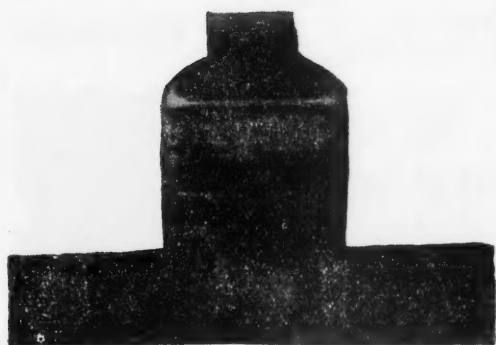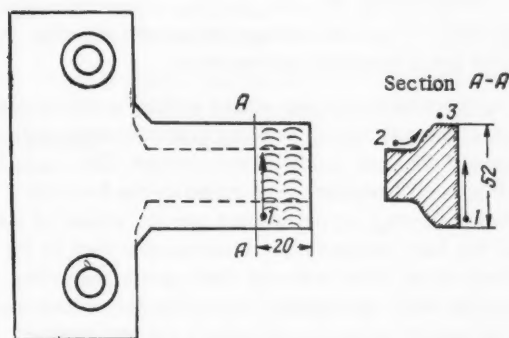
Fig. 2. Outside view of a naturally worn punch.

The working part cf the punch was ground on a grinding machine (rounding-off 1.5°). The tests were made on slot cutting in small and large flanges of interstitial and butt backings. The table gives the durability (number of slots cut) of individual punches made of steel 5KhNV when new and reconditioned, the output of the punching machine being 3,000 rail backings per shift. For comparison, the durability of punches made of steel U10 is given, the output of the punching machine being in this case 1,500 rail backings per shift.

It is seen from the table that the reconditioned punches made of steel 5KhNV cut 15,000 to 37,000 slots in one setting in the punching machine and their total life after three to five weldings constituted 60,000–104,000 slots. The reconditioned punches made of steel U10 cut only 300–600 slots. Total durability of these punches after five weldings was 2000 slots.

Durability of Punches Made of Steel 5KhNV and U10

Fig. 3. Method of welding-on of the cutting part of the punch.

Steel	Durability of punches in one setting, No. of slots						
		after welding-on					
	New	first	second	third	fourth	fifth	Total
5KhNV	700	37410	21880	21500	—	—	81500
	800	26630	23350	27727	5000*	21010	104510
	900	15320	2578*	20510	8500*	10250	60050
	1000	24862	—	—	—	—	25862
	1000	21000	—	—	—	—	22000
	1000	15000	—	—	—	—	16000
	1000	29236	—	—	—	—	30236
U10	83	600	417	400	300	290	1990
	100	600	400	502	250	200	2052

*Punches were welded by unskilled electrowelders.

The durability of reconditioned punches depends in the first place on the quality of the weld and on the mechanical and thermal treatment and not on the number of times they were reconditioned.

Characteristic wear of a well reconditioned punch is shown in Fig. 4. The cutting edges, welded with high-speed steel, proved to be so durable that the foundation metal of the punch face was worn but the edges were still preserved.

On the punches which were badly welded-on, fatique cracks appeared in the course of operation in the places where a rounded part passes into a square one

Fig. 4. Wear of correctly welded-on punches.

and the edges frequently crumbled off. As a result of rough machining of the punch its cutting part breaks off along the fatigue cracks which appear because of the concentration of stresses around the lines left by the cutting knife. If corner rounding-off is not strictly observed the puncher is quickly rendered useless (crumbling of corners, break-off etc.)

RAGGING OF ROLL SURFACE BY KNURLING

A.I. Solovyev

Senior **Roll Designer of Makeevka** Metallurgical Works

Roll ragging in various forms of groove cutting or welding-on is frequently applied in order to facilitate the gripping of metal by the rolls. However, shallow and sparse ragging or thin welding-on do not improve the gripping conditions substantially. Wide and deep ragging or a high welding-on facilitates the gripping considerably but results in surface defects of the rolled product. Moreover, the welding places become hardened and are difficult to machine when the rolls are reconditioned.

A new method of improving the gripping of metal by the rolls—ragging of passes by knurling—is being adopted in the rolling mills of the **Makeevka Works.** A thick checker of shallow rectangular pyramids (Fig. 2) is pressed out on the roll lathe with a special knurling toothed roller (Fig. 1).

The roller, of 150 mm diameter and 28 mm working width, is mounted in a special housing on roller bearings. The roller is made of tool steel (U10, U12, 35KhVS) and is quenched and tempered.
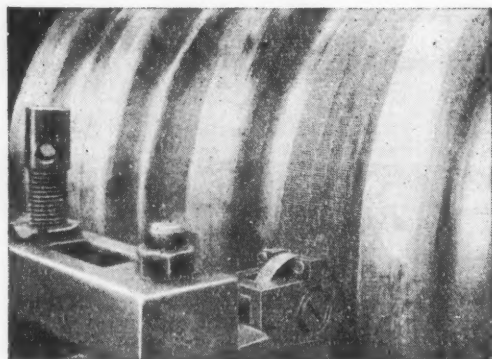
Fig. 1. Ragging of blooming mill rolls on the roll lathe.

The knurling of one stretch is done in 4-5 revolutions of the roll, the roller being gradually tightened by means of the usual handle of the carriage. The knurled surface is oil lubricated. The pyramids can be made either projecting out or indented into the surface of the roll. We have adopted indentation because then on the surface of the rolled material there appear projecting pyramids which subsequently are rolled out without trace in the smooth passes. On the other hand, the indented pyramids on the rolled material or projecting, closed, sturdy ridges, and the traces of ragging may remain on the finished product.

The ragging of the roll surface by knurling has several advantages compared with groove cutting or welding-on.

1. The operating efficiency of the mill increases through eliminating the stoppages due to slippages on gripping, especially after the roll changing. With certain forms of ragging and conditions of rolling, the angle of bite may exceed 34°, which is considered the maximum with deep grooving.

2. Jamming of billets in the mills with constant high roll speed, as well as the jerks occurring in the rolling train if the rolls are grooved, are eliminated.

3. The surface quality of the finished product is improved.

4. The life of rolls is increased as their surface is work-hardened by knurling.

The ragged surface becomes gradually worn, the upper roll being worn faster (effect of scale), but a roughness and a pattern of fissures (Fig. 3) determined by the knurling are formed on the surface and hence the gripping of metal by the rolls does not deteriorate.
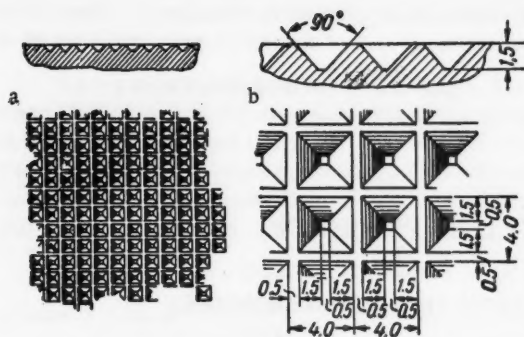
Fig. 2. Surface of a ragged roll: a) actual size;
b) enlarged (x 4).

The ragging by knurling has been applied on rolls in the blooming mill and on the first stand "A" of the continuous billet mill 630. The six-month operation has produced satisfactory results. Initial tests are conducted now on the reducing stands of the billet and large-section mills 600 and testing of ragged rolls on the roughing stands of section mills 850, 330 and 280 is contemplated.

Fig. 3. Lead imprint of blooming mill roll surface: a) before use; b) after 10 days work; c) after 20 days work.

Further tests of various forms and dimensions of ragging for different conditions of rolling, and checking of roll service and the quality of product are required for the improvement and general adoption of the method of roll ragging by knurling.

SCHOOLS OF ADVANCED EXPERIENCE

SMELTING OF LOW-MANGANESE IRON IN SOUTHERN WORKS

Candidates of Tech. Sci. V.P. Onoprienko and B.N. Starshinov,

Engineers N.M Kharchenko and A.A. Baby

During the smelting of conversion pig iron, containing 1.5 – 3.5% Mn, approximately 60–70% of valuable manganese which, with the charge, enters the blast furnace, is lost in the slag and blast-furnace dust and only 30–40% goes into the pig iron. According to the data of the Dzerzhinsky Works, up to 75% Mn contained in pig iron, is removed with the primary slag from the open-hearth furnace. Altogether 70–80% of manganese delivered to the works is lost with the slag.

Manganese in open-hearth pig iron improves the desulphurization process outside the blast furnace. Investigations showed that the manganese content in pig iron had a noticeable effect on the sulfur content in the end product only if there was more than 2.0–2.5% manganese. The desulphurization does not occur if the manganese content is small.

The conditions of smelting in the blast furnace have an effect on the sulfur content of pig iron. For the lowering of the sulfur content of pig iron to the level specified by GOST or according to the specific technical requirements it is necessary, when the manganese content is decreased, to increase basicity of the slag and the heating of the hearth. Under such conditions the utilization of manganese in the blast furnace will be improved and the MnO content in slag will decrease. However the decrease of the MnO content in slag will be greater than it would be if there were the same reduction of Mn content in pig iron without the change of the operating parameters of the furnace. Hence for imparting adequate fluidity to the slag at an increased basicity, the MgO and Al_2O_3 content in slag should be increased by an amount greater than the reduction in MnO content.

From 1940 – 1941, pig iron for Bessemer converters in the Dzerzhinsky Works was made with magnesia slags; increasing the MgO content in the slag made it possible to obtain stable slags and a satisfactory low sulfur content in pig iron with a smooth and steady blast furnace operation. Pig iron for Bessemer converters is now similar to that for open-hearth furnaces in its silicon and sulphur content but the manganese content in Bessemer pig iron is considerably lower. The experience in pig iron production for Bessemer converters under the conditions prevailing in the South of the USSR showed that it is quite possible to obtain low-manganese pig iron for open-hearth furnaces with an adequately low sulfur content.

Since March 1954, one of the blast furnaces of the Dzerzhinsky Works has been producing open-hearth pig iron conforming to GOST 805-49. Manganese ore (grade III and IV) of the Nikopol deposits is introduced into the charge. First tests in making open-hearth, low-manganese pig iron were made in September-October 1954. At that time the attempts to produce low-manganese pig iron with a steady normal sulfur content were

unsuccessful. On reduction of the manganese content in pig iron to 1.1%, the MgO content in slag increased to 4-5% the basicity remaining at 1.0—1.1; the sulfur content increased to 0.1%.

During experiments from April 24 to May 8, 1955, the basicity of the slag was increased to 1.25 and the mean MgO content in the slag constituted 4.73%. Manganese content in pig iron decreased to 1.38% (to 1.12% in some cases), and MnO content in slag decreased to 1.82% (1.44% on some days). The ratio of the sum $CaO + MgO + MnO$ to silica constituted 1.43%. The sulfur content in pig iron was, nevertheless, high and unsteady (0.051% mean content over the trial period). The minimum sulfur content of pig iron was observed when there was maximum MgO content in the slag.

At the end of July 1955, the slag basicity was increased, to 1.30—1.35. MgO content in the slag was increased to 5.5% at $\dfrac{CaO + MgO + MnO}{SiO_2}$ = 1.45—1.50. A steady manganese content in pig iron was attained at a level of 0.80—0.85% and with a steady and relatively low (0.045%) sulfur content.

On comparing the operation indices of this furnace of the Dzerzhinsky Works to the production of the conventional open-hearth pig iron (February – April 1955) with the indices on low-manganese pig iron production, the reduction in manganese ore consumption by 80% (iron content in the charge increased by 5.6%) and coke consumption by 4.2% can be noted. Furnace output increased by 4.2%, slag yield decreased by 19.2% and the CO_2 content in the flue gases increased by 1.15%.

Good furnace operation indices were obtained at the following main parameters of operating conditions:

Slag basicity	1.28—1.30
$\dfrac{CaO + MgO + MnO}{SiO_2}$	1.45—1.46
MgO content in slag, %	5.5 —6.5
same, Al_2O_3, %	5.5
Blast temperature, °C	750—800
Blast humidity g/m^3	25—28
CO_2 content in flue gas, %	
at the periphery	8— 9
at the half-radius distance	16—17
at the furnace centre	9—10

During the smelting of low-manganese pig iron it is especially important to maintain smooth operation and uniform heating of the blast furnace. This depends to a large extent on the steady uniform quality of raw materials. Variations in raw material composition have an effect on pig iron composition; they cause an increased sulfur content in individual batches such as occurred in the Dzerzhinsky Works where the sulfur content in pig iron was 0.080% or more in some batches.

Low-manganese pig iron may take up sulfur on the way from the blast furnace to the consumer. This process is enhanced by poor cleaning of ladles and by the treatment of pig iron in ladles with coke fines. Pig iron, when entering the mixer, contains 0.005—0.010% sulfur more than when it is collected from the blast furnace.

Sulfur saturation can also take place in the mixer if the minimum amount of metal is not maintained, the slag is not run off and graphite is not removed.

Low-manganese, conversion pig iron has a higher temperature than pig iron produced in the ordinary way. On mixing it with the ordinary pig iron in the mixer the temperature of the latter increases and the manganese content decreases, thus causing sulfur transfer from slag to metal.

At the "Zaporozhstal" Works, from July to December 1955, the manganese content in pig iron in furnace No.1. was reduced from 2.5 to 1.8-2.0%, the ratio $\dfrac{CaO}{SiO_2}$ in the slag being simultaneously increased to 1.27, at 4.5% MgO and 8.5—10.5% Al_2O_3 content in the slag.

At these Works, as distinct from others, slags have an increased Al_2O_3 content and hence slag mobility is high. Therefore a low MgO content can be maintained at an increased slag basicity.

With 1.8−1.93% manganese content in pig iron, sulfur content consitututed 0.043−0.045%.

On reducing MgO content in slag to 3.5% or less and increasing basicity $\dfrac{RO}{SiO_2}$ to 1.46, the manganese content in iron decreased to 1.66% but the furnace operating deteriorated, sulphur content in iron increased as the rate of smelting decreased. This shows that in these cases slags lost their mobility and desulfurizing power.

During the making of low-manganese pig iron at "Zaporozhstal" Works the slag basicity was increased but the manganese ore consumption decreased only by 35% and the corresponding slag amount increased by 6.0%. Consequently, the furnace output decreased by 1.5% and coke consumption increased by 1.5%. Thus no advantage was gained. To get a positive effect it is necessary to reduce manganese content in pig iron to 1.0% or less.

As a result of the change-over to the production of pig iron with a reduced manganese content at Kirov Works in Makeevka, the furnace output increased by 1.8−4.7%, manganese ore consumption decreased by 23−59% and slag yield fell by 5%. The coke consumption in furnace No. 1. remained practically the same and in firnace No. 4 decreased by 3.4%. Pig iron cost decreased by 2.3−2.6%.

The sulfur content of the iron was nevertheless high, the reason being a low basicity of the slag and its low MgO content.

The plant then started production of pig iron with 0.91−1.05% manganese content. The manganese ore was eliminated from the charge, only the sinter with 0.99% manganese content being employed.

The basicity of the slag was increased from 1.25−1.28 to 130−131, the ratio $\dfrac{CaO + MgO + MnO}{SiO_2}$ in furnace No. 2 was raised from 1.38 to 1.41 and in furnaces Nos. 1 and 4 was maintained at 1.42−1.43.

As a result of eliminating the manganese ore, the iron content in the charge increased by 0.7−0.8%, slag yield in furnaces No. 1 and 2 decreased by 5.6−6.0% and in furnace No. 4 by 12.3%. Coke consumption in furnace No. 4 decreased by 5.7%, and in furnace No. 1 it remained unchanged; the output of furnace No. 1 decreased by 0.7%, of furnaces Nos. 2 and 4 increased by 6 and 5% respectively.

On the complete elimination of manganese ore from the sinter charge the manganese content in pig iron decreased to 0.17−0.40%. In spite of an increase of silicon content in pig iron from 0.63−0.68% to 0.95−1.11% (in No. 2 and No. 4 furnaces), the sulfur content increased in furnace No. 2 from 0.043 to 0.050%, and in furnace No. 4 from 0.028 to 0.039% but decreased in furnace No. 1 by 0.003%.

Large fluctuations of silicon and sulfur content in iron during the experimental heats, these fluctuations being caused by the variations in the quality of raw materials, especially limestone, should be mentioned. The sulfur content in iron was maintained within the limits of 0.020−0.045% and increased only when a marked deterioration in the quality of the flux occurred.

As a result of eliminating the ore, grade 20, containing 3-4% MgO, from the charge, the MgO content in the slag, even at an increased consumption of dolomitized limestone, decreased from 4.22−4.30 to 3.2-3.9%. This also had an effect on the sulfur content in pig iron.

It follows from the above that prior to the change-over to the production of iron with a manganese content of less than 1.0% it is necessary to improve the ore preparation.

INCREASING THE DURABILITY OF CRANE WHEELS*

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(Dzerzhinsky Works)

The inadequate durability of crane wheels is due to the constructional faults of the wheels themselves and of undercrane tracks, as well as to the factors related to the material used for crane wheel manufacture, the method of casting and forging, and the conditions of thermal treatment.

The Dzerzhinsky and "Zaporozhstal" Works, Nizhne-Tagil Metallurgical Combine, and other plants manufacture rimming wheels by casting. In the past the wheels made in Dzerzhinsky Works were cast of steel 45L or 50L. After casting, the wheels were annealed according to a generally accepted method, machined on the lathe, and finally the rolling surface was hardened with an oxy-acetylene flame on the universal hardening machine (Fig. 1) by means of a conventional welding torch with a nozzle of Dzerzhinsky Works design.
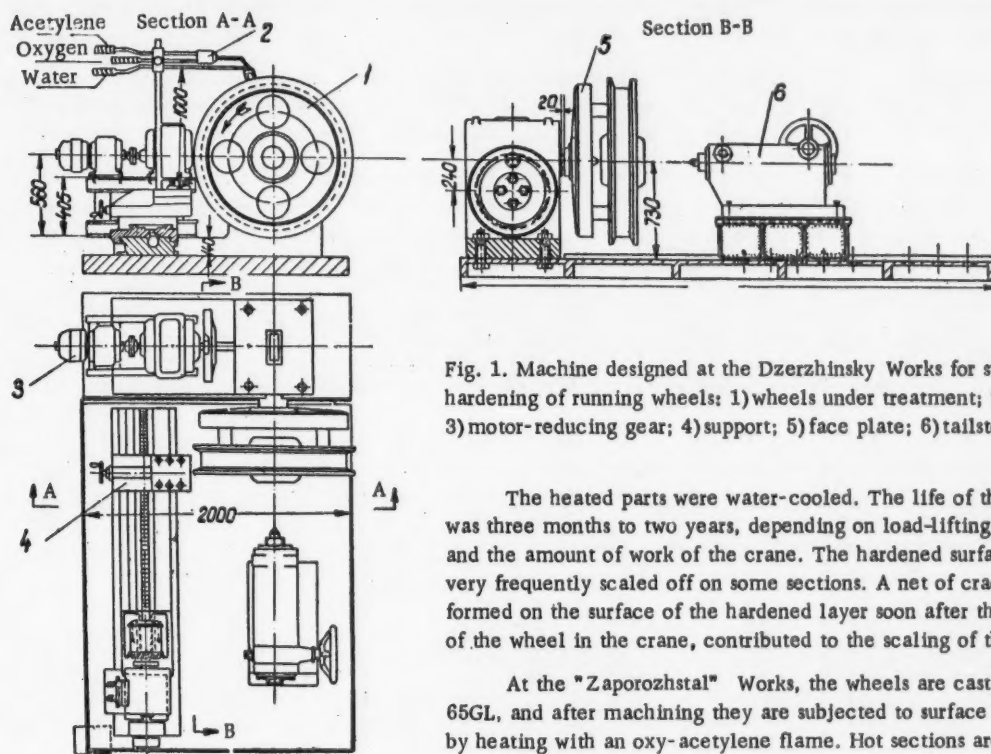
Fig. 1. Machine designed at the Dzerzhinsky Works for surface hardening of running wheels: 1) wheels under treatment; 2) torch; 3) motor-reducing gear; 4) support; 5) face plate; 6) tailstock.

The heated parts were water-cooled. The life of the wheels was three months to two years, depending on load-lifting capacity and the amount of work of the crane. The hardened surface layer very frequently scaled off on some sections. A net of cracks which formed on the surface of the hardened layer soon after the mounting of the wheel in the crane, contributed to the scaling of the layers.

At the "Zaporozhstal" Works, the wheels are cast of steel 65GL, and after machining they are subjected to surface hardening by heating with an oxy-acetylene flame. Hot sections are cooled with compressed air. The life of wheels prepared in this manner varies between two months and two years.

At the "Azovstal" Works, the hardening of the crane wheel runway with high-frequency current is presently being introduced; there are however reasons to believe that even this method will not ensure a high durability of the wheels.

At the Kirov Works in Makeevka and at the Magnitogorsk Metallurgical Combine compound wheels are made with forged tires of steel 50, 65 and 65G.

At the Kirov Works, after machining, the rolling surface is subjected to surface hardening with coke oven gas flame, and at the Magnitogorsk Combine the surface is normalized according to the generally accepted methods for the given steel grade. The tire durability is unsatisfactory at both works. There are frequent cases of tire flattening (spreading) in the course of operation. The short life of the wheels can be explained by the fact that the depth of the hardened layer obtained by surface hardening is limited and the transition from the hardened surface layer to the untreated core is rather sharp. Furthermore, the measurements of the temperature at the surface and inside the cross section of the treated parts is difficult because of the specific conditions of the gas-oxygen hardening. The temperature can be controlled by changing the basic variables of the hardening process (the speed of the hardened surface and the distance between the burner and the hardened surface) as well as the variables of the flame. Small deviations of these variables, very frequently difficult to allow for, may cause an overheating of the surface layer resulting in the formation of small cracks and metal scaling from the surface. This is especially apparent on the parts which are subjected to large cyclic loads in the course of operation. The running wheels of heavily loaded overhead electric cranes belong to this category of equipment.

Attempts to modify the conditions of thermal treatment were not successful;— the durability of the wheels did not improve and the surface layer scaled as before.

Consequently, at the Dzerzhinsky Works a new method of wheel manufacture was adopted: an experimental batch of wheels was cast of medium manganese content steel 50G2L and was subjected to normalizing with a subsequent high temperature tempering. This decision was based on the following reasons.

The imported cranes in the open-hearth plant of the Dzerzhinsky Works and in the new mechanical shop DZMO have running wheels made of steel whose chemical composition is very similar to the composition of steel 50G2; the wheels have been in operation for a long time without any apparent signs of wear. Judging by the microstructure of these wheels, they were, apparently, normalized in the final thermal treatment. The hardness of their working surface is $260-280H_B$.
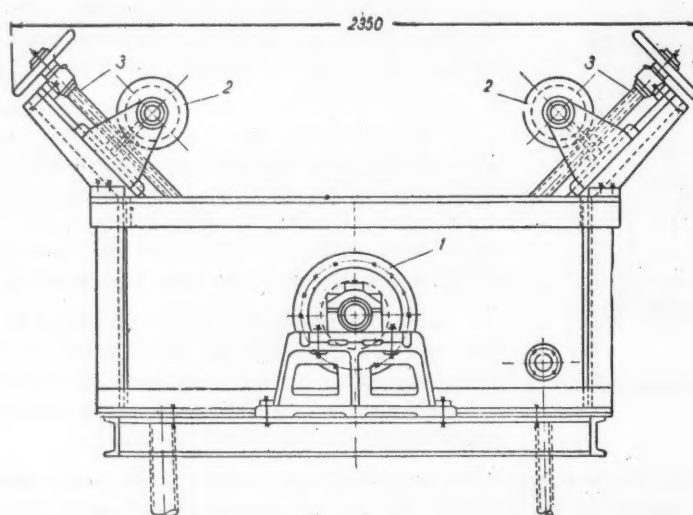
Fig. 2. Diagram of the equipment for the sorbitization of running wheels,
designed by KMK (Kuznetsk Metallurgical Combine): 1) driving roll;
2) idle rolls; 3) guide.

The life of wheels subjected to thermal treatment according to the new method, increased, but not greatly;— on the average by 20—30%. The wheels were breaking down mainly because of wearing away at the flanges.
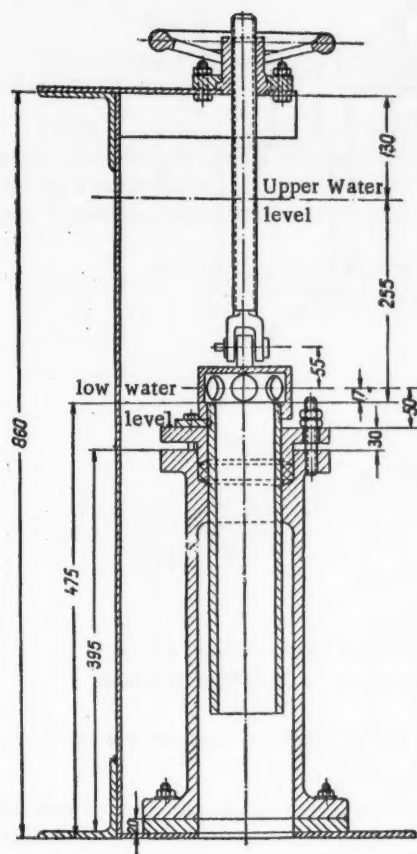
Fig. 3. Water-level regulator in the quenching tank.

At the Kuznetsk Metallurgical Combine the running wheels are cast of steel 50G2L and are subjected to sorbitization in a special machine (Fig. 2).

The crane wheel heated in a furnace to a temperature above A_{C_3} is transported by means of a crane with a C-shaped hook into the quenching bath and is placed on supporting rolls. The roll (1) is driving and the rolls (2) are free-running.

The driving roll and the part resting on it are rotated by an electric motor via reducing gear. The free running rolls together with their mountings can be moved downward along the inclined guides (3) and are placed in position depending on the diameter of the part under treatment. The water level in the quenching bath is controlled by means of a special device situated above the overflow pipe welded to the bottom of the bath (Fig. 3).

The water level in the tank is adjusted depending on the size of the wheels. Usually, the water level is maintained at 50 mm above the surface of the supporting driving roll. The use of the C-shaped hook allows the placing of wheels onto the running rolls without stopping the machine. The durability of the wheels treated in this manner exceeds by two to three times the durability of other wheels working under similar conditions at other works.

A sorbitization machine (Fig. 4) in which some design defects of the Kuznetsk equipment are rectified, has been designed at the Dzerzhinsky Works.

The wheel, heated to a temperature above A_{C_3}, is gripped between the cones of the machine and is rotated by the electric motor. The speed can be smoothly controlled by a variable gear. There is a controlling device for the water level in the tank (Fig. 3). Wheels of 650—1150 mm outside diameter can be thermally treated with this equipment. The working surface of the wheel should be immersed in the water to a depth of 40—50 mm.

The hardness and the depth of the quenched layer is controlled by varying the depth of immersion of the wheel or the time of its rotation.

The crane wheels are made of steel 50G2 of the following chemical composition, %:

C	Mn	Si	P	S
0.44—0.55	1.4—1.8	0.17—0.30	≤0.040	≤0.045

After casting, the wheels are annealed at 780—800°C. The time of exposure to the temperature of annealing depends on the size of the wheel. The cooling takes place at a rate not higher than 60°C per hour; the hardness after annealing is not more than 229 H_B. The wheels are then machined including preliminary boring of the axial hole. The allowance for the deformation during the thermal treatment is made in the axial bore. The experience of the Kuznetsk Metallurgical Combine has shown that for the axial opening of 70—100 mm the allowance constitutes 5 mm and for the axial opening of 100—160 mm diameter the allowance is 8 mm.

After machining, the wheels are subjected to quenching with self-tempering. The wheels are quenched from 780—800°C. Only the working surfaces of the wheels are immersed in water in the sorbitization machine. The time for one revolution of the wheel depends on its outside diameter and varies between the limits of 50 to 300 sec. The wheels are then placed on the plant floor for tempering which takes place due to the heat contained by the bulk of the wheel. Hardness of the wheels after the termal treatment should be 280—360 H_B.
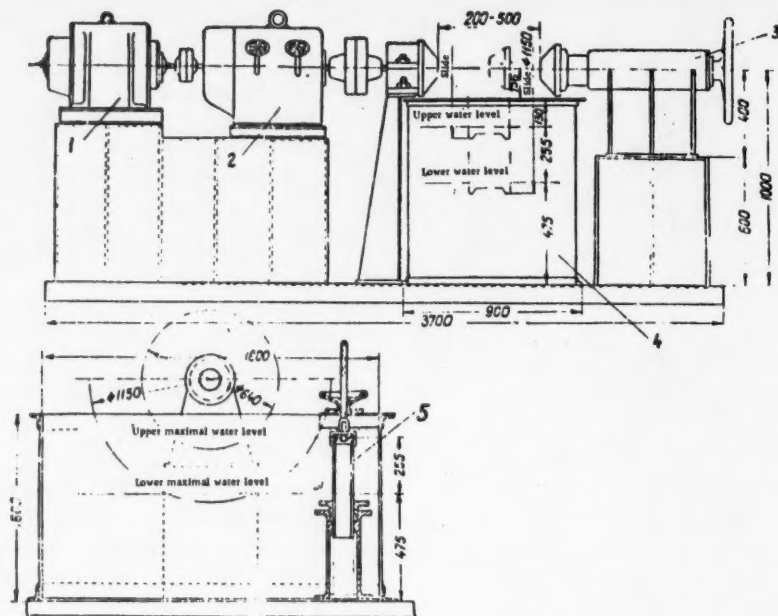
Fig. 4. Equipment designed by the Dzerzhinsky Works for the sorbitization of wheels:
1) electric motor; 2) variable gear; 3) tailstock; 4) tempering tank; 5) level regulator.

After the thermal treatment the wheels are finally machined and the surplus of material allowed for the thermal deformation in the axial hole is removed.

The running wheels of the cranes (teeming and casting) transporting liquid metal, are made of steel 50G2, but instead of quenching and tempering they are normalized at 810—830°C with a subsequent cooling in calm air. The hardness of the running wheels after the thermal treatment should be not less than 269 H_B.

Such thermal treatment of the overhead electric crane wheels extends their life considerably. The adoption of this method will enable other works to save substantial quantities of metal.

However, the method of wheel durability improvement described above is not the only one. The organization of centralized manufacture of seemless rolled crane wheels for all the metallurgical works seems to us more advantageous.

The rolling of the wheels will undoubtedly improve their structure and mechanical properties thus increasing their service life. In order to simplify the solution of this problem all metallurgical and machine works, in collaboration with a design organization e.g., Gipromez, should standardize the wheel dimensions.

WATER-COOLED PLATES OF THE TAPPING SPOUT

Water-cooled plates (see Figure) were fitted into the tapping spouts at the open-hearth plant of the Ufaleisk Metallurgical Works with a view to improving sanitary working conditions.

The plate, made of 10-mm sheet steel, is in the form of an open compartment 1300 × 1000 × 70 mm. Cooling water flows by gravity from tanks through 32 mm diameter pipes. The plate capacity is 84 liters. The plates are installed in sections along the sides of the spout; each section consists of four consecutively-joined plates. Continuous water circulation inside the plates ensures rapid cooling of spout walls.
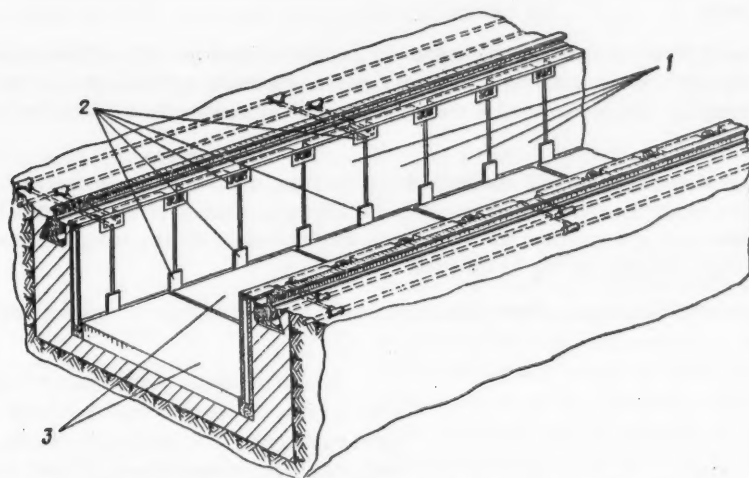
Water-cooled plates in the tapping spout: 1)plates; 2)protective facing; 3)bottom plates.

Working conditions during the dismantling and assembling of the spout have improved considerably since the introduction of the water-cooled plates.

Presently, walls of regenerators and slag chambers are also being provided with water-cooled plates in order to eliminate heat radiation.

V. Ivanov.

(Ufaleisk Metallurgical Works)

STANDARDIZATION OF PARTS MADE OF STEEL G13L

G.I. Shandrenko and Yu. M. Ponomarev

All-Union Scientific Research Institute for Ferrous Metals

Manganese steel G13L is widely used in ferrous metallurgy for the manufacture of spare parts and components of equipment in which abrasion takes place during its operation. Such parts are used in iron and steel works, by-product coke plants, refractories plants and mines.

As a rule the manganese steel parts are cast at the plants in their casting shops for the plant's own use. The standardization of the product is insignificant and the technology and the mechanization of casting processes are at a low level.

The unification of these parts will make possible the organizations of specialized production, simplify the nomenclature of components in use, increase the size of batches, ensure the application of serial methods of operation and, consequently, will make possible the mechanization of molding by the introduction of pneumatic molding machines.

The application of mechanical mold making reduces the time of ramming by 15 times (when operating with pneumatic jolt-ramming machines) compared with manual mold making, eliminates time losses in the adjustment of defective spots and improves the accuracy in the dimensions of the casting owing to the elimination of possible breakage of pattern during its removal.

The accuracy of the dimensions is especially important in the case of high-manganese steel which is very difficult to machine.

Fig. 1. Diagram of the armor plating of the ore-chute at the "Azovstal" Works.

Protective armor plates made of high-manganese steel prevent the wear of important units in metallurgical equipment. The dimensions of these plates vary, each size being manufactured in small quantities only. For instance, the coke hoppers of the blast furnaces No. 3 and No. 4 at the Makeevka Metallurgical Works are lined with plates of 38 different sizes. Of these, for 9 sizes two to five plates are made of each size, and for the remaining 29 sizes only one plate is cast for each size. The chute for ore and sinter at the blast furnace of this Works is lined with plates of 32 different sizes; two plates of each size are made for 6 sizes, and for the remaining 26 sizes, one plate only is made of each size.

It is seen from the above how important it is for specialization and cooperation in the production of high-manganese steel castings to introduce standardization of armor plates and other components.

For the reduction of the sizes and types it is necessary to standardize length, width, thickness and the form of distribution of fastening holes, etc.

The thickness of the plates in the coke scale hopper of the blast furnaces at the "Azvostal" Works was increased to 35—40 mm and this extended their life 2—2.5 years. In the ore chutes at those works, armor plates of 40 mm thickness in the upper part and 60 mm in the lower part were applied. The life of the plates increased also to 2—2.5 years. The inside surface of the
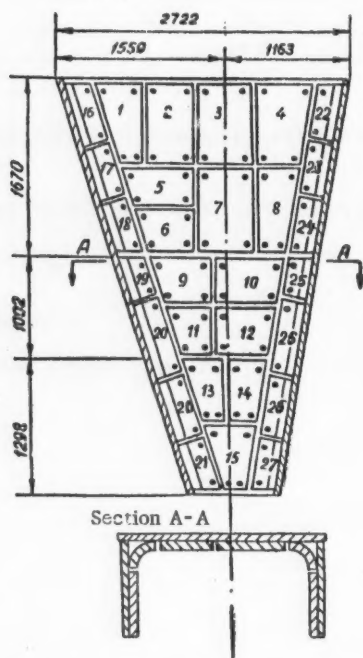
blast-furnace skip was lined with 30-mm thick, high-manganese steel plates; this increased the skip life from 3—4 to 8—14 months. Increasing the thickness of the plates in the receiving funnel of the charging equipment of the blast furnace increased its life from 3—4 months to 2.5—3 years.

The life of the revolving charge distributor funnels increased from 6 months to 2 years by lining them with 25-mm thick plates in two layers in the upper part and three layers in the lower part. The increase in the thickness of the guard rings of the small bell rods from 40 mm to 80 and 120 mm increased the life of the rings from 5 months to 2 years.

Taking into account the experience of "Azovstal" Works, one may accept the following standardized thickness of the armor plates: 30, 40, 60 and 80 mm.

Other linear dimensions of the plate are also very diverse. This variation is not always a matter of necessity. More frequently it is derived from underestimation (by our designers) of the standardization problems, the solution of which would facilitate the setting up of specialized production.

It is seen from the diagram of the armor plating of the ore-chute bottom at the "Azovstal" Works (Fig. 1) that there are 27 plates of various sizes out of which only two plates (positions 20 and 26) are of the same size, the remaining plates being all different sizes.

A unified system of linear dimensions of armor plates for this chute is shown in Fig. 2. The number of sizes is reduced to five. Five plates for each position 1, 2, 4 and 5 are required and ten plates are required for the position 3.
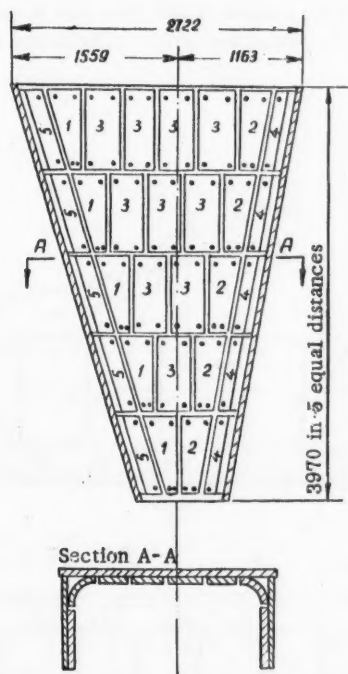
Fig. 2. Diagram of the armor plating of the ore-chute with the unified plates.

If special corner plates at the junction of the chute bottom with walls are dispensed with (Fig. 3), the scheme can be still further simplified. The chutes at Kirov and Voroshilov Works and the majority of the blast furnaces of the Dzerzhinsky Works are lined according to such a scheme. It allows the reduction of the plate sizes by two more compared with the diagram in Fig. 2, and the reduction of the total number of plates from 30 to 20.

At the Voroshilov Works the upper limit of the armor plates in the skip (10 cu/m volume) is inclined relative to the lower wall of the skip (Fig. 4), the intention being to reduce the weight of lining. Such an arrangement considerably increased the number of plate sizes, the expenditure on the patterns and losses due to the production of 4 extra sizes (positions 7, 8, 10, 11) reducing the metal saving to nil. If the upper line of the plates were arranged parallel to the lower wall of the skip, then only one size instead of the four would be required (Fig. 4,b).

Dispensing with the corner plates at the joints of the bottom and side walls (positions 5, 6, 7, 8, Fig. 4,b) will still further reduce the number of plate sizes.

Countersunk and cheese-head bolts of different diameters are used for fixing the plates even in one and the same machinery. When the standardization of armor plates is carried out, bolts of one or two sizes with countersunk head (Fig. 5) should be accepted because cracks appear frequently in the corners of square holes (for the square bolt heads) on casting or in the course of equipment operation. Furthermore, round sunk holes make it possible to make the best use of the plate thickness and to extend the plate life compared with other methods of fixing the plates of the same thickness.
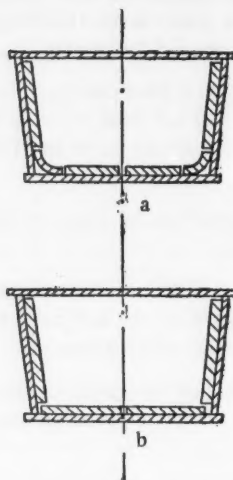
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Fig. 3. Diagram of lining the ore
chute: a) with corner plates;
b) without corner plates.

The unification of fixing arrangement will result in the reduction of the number of bolt sizes now in use in the industry and the reduction of the bolt equipment. It will facilitate the interchangeability of bolts and will establish conditions for an extensive mechanization of bolt production.
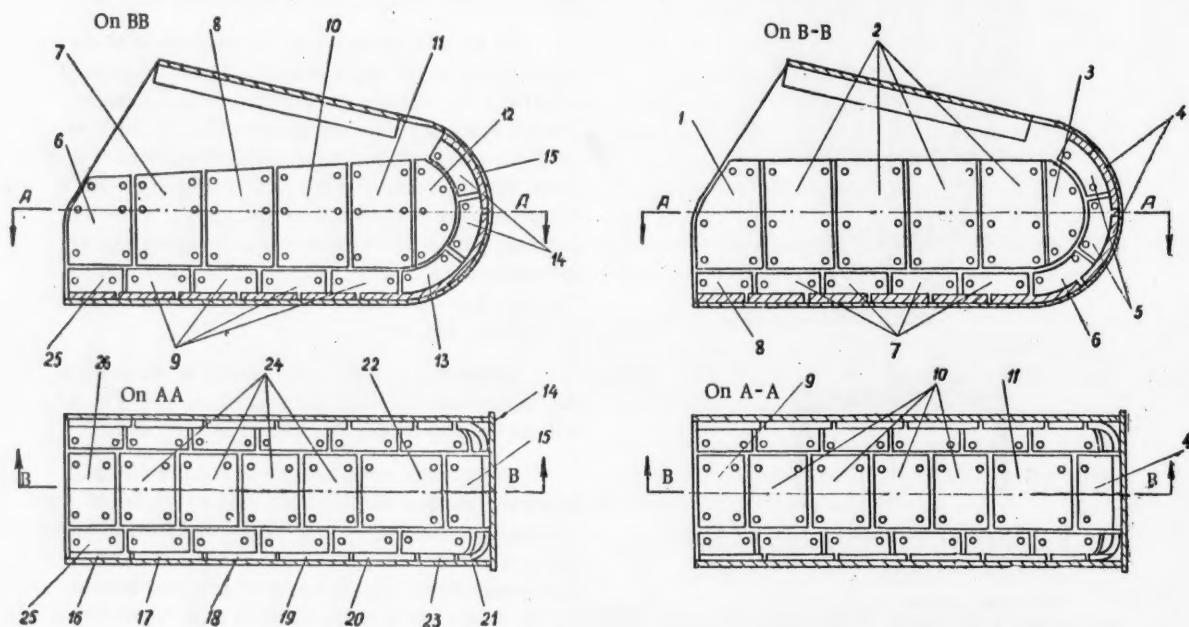

Fig. 4. Diagram of the armor plating of the internal surface of 10 cu m skip (blast furnace No. 3 of the Voroshilov Works). On the left: present; on the right: proposed.

The above given examples do not exhaust all the possibilities for the unification of armor plates and of other parts made of high-manganese steel, and they show how wide is the scope for unification and how important it is for the organization of specialized and coordinated production of these components.
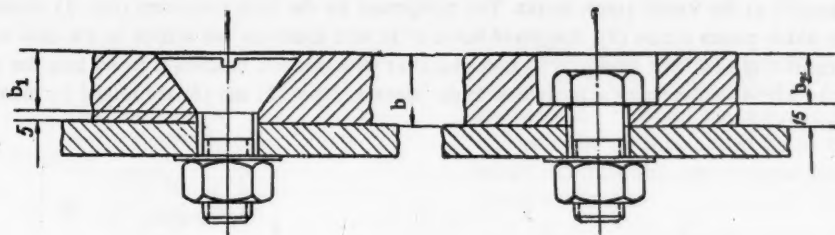
Fig. 5. Methods of the attachment of armor plates.

UTILIZATION OF FLUE GASES FROM THE OPEN-HEARTH FURNACE
FOR HEAT TREATMENT OF METALS

The heat of the flue gases from the open-hearth furnace is made use of in the heat treatment of ferrous alloys and other materials at the Verkh-Isetsk Works. The equipment for the heat treatment (Fig. 1) consists of a bunker (1) through which passes a tube (2). Staggered holes of 10 mm diameter are drilled in the tube wall. An ejector (Fig. 2) consisting of a bent tubing of 25 mm diameter with one end blinded is fitted into the tube (5). Compressed air at 3.5—4 atm pressure is delivered to the ejector. Pipes (2) and (5) are joined by means of a connecting piece (4).
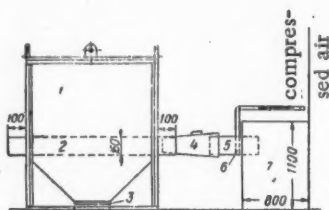
Fig. 1. Equipment for the heat treatment of materials: 1) bunker; 2) tube; 3) outlet for the removal of material; 4) connecting piece; 6) chamber tube; 6) ejector; 7) chamber at the wall of the vertical duct.

Fig. 2. Ejector fitted into the tube.

During the drying, the outlet of the tube is blanked off and compressed air is let into the ejector. The compressed stream of air draws in the hot gases from the vertical flue of the furnace, the gases pass through the holes in the tube and penetrate through all the material in the bunker. The temperature of the air — flue gas mixture is about 1000°C. The roasting of the material in the bunker (2600 kg) is continued for 1—1.5 hrs, most of the material being brought to red heat.

Such equipment has been operating at each furnace in the plant since January 1, 1957. As a result of roasting of the ferrosilicon for the alloying of steel, cases of flaking during charging of ferrosilicon into the trough have ceased, the gas content of metal has decreased, ans so has the rising of ingot.

The equipment is simple and reliable in operation. The cost of installation is small.

A.B. Dokshitsky

(Verkh-Isetsk Metallurgical Works)

ROLLING BEARINGS IN OVERHEAD CRANES

L.K. Kuznetsov

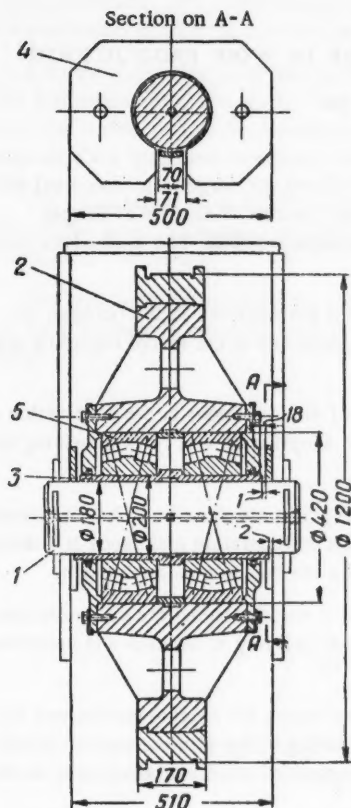Engineer-Designer of the Design Department of the Chelyabinsk Metallurgical Works

Most of the overhead cranes in metallurgical plants are equipped with sliding bearings. The crane mechanisms operate under medium and heavy loads. The large amount of dust in the air accelerates the wearing off of bearing bushes. All this increases the consumption of antifrictional materials.

The machinery of the overhead cranes at the Chelyabinsk Metallurgical Works is being converted to rolling bearings.

The figure shows one variant of the bearing conversion from sliding to rolling bearings of the running wheels of the 140/30 ton pouring crane. The axle (1) of the running wheel (2) and its mounting in the balance box are preserved in the original form. The assembled unit is introduced into the balance box from below. The axle (1) is then introduced and is secured with the axle holders. The hub (3) has a fin which fits into a groove in the plate (4) preventing the rotation of the hub. An axial displacement of the wheel is allowed for by the clearance between the hub (3) and the plate (4).

The spherical roller bearings (5) (No. 3640) do not require adjustment in the course of operation and allow for an axial loading up to 20% of unused radial loading.

The application of rolling bearings instead of sliding bearings results in the reduction of energy consumption on the overcoming of frictional resistance by 20–30%, the maintenance of the equipment is facilitated, the consumption of lubricants is lowered and the operating periods between the necessary overhauls are extended.

Open gear transmissions mounted in the balance box and in the supporting girder of the overhead pouring crane are also being converted to rolling bearings, the existing axles being retained.

Figure. Arrangement of rolling bearings in the running wheel of the pouring crane: 1) axle; 2) wheel; 3) hub; 4) plate; 5) spherical rolling bearing.

The operating experience of several overhead cranes confirms the expediency of conversion of their rotational mechanisms and, above all, of the running wheels, to rolling bearings.

NEW BOOKS

INTER-WORKS SCHOOL OF ADVANCED PRACTICES IN WIRE PRODUCTION

Moskow, Metallurgizdat, 1957, 95 pp.

An inter-works course of advanced practices in the production of 0.2 mm and larger diameter steel and iron wire was arranged in December 1955 — January 1956 by the Scientific Technical Society of Ferrous Metallurgy in cooperation with Glavmetiz MChM (Ministry of Ferrous Metallurgy) SSSR. The work of the course is presented in the above booklet.

The methods of metal surface preparation for drawing at seven works are discussed in the booklet, the advantages and disadvantages of each method are noted and the recommendations of the school regarding the improvement of the process are given.

Special attention is paid to the introduction of the mechanization of the operations in the preparation of metal for drawing, the improvements in washing and drying of metal, the adoption of wire rod Parkerizing after the pickling, and other problems.

Next the methods of manufacture and treatment of wire at different plants are described. A new electrolytic treatment of circular wire is recommended. By the adoption of this method the operating efficiency is quadrupled. Mechanical sharpening of needles for working wire and a new shape for the die hole are recommended.

The school examined the methods of wire drawing at various works. It recommended the modernization and standardization of the wire drawing equipment, the mechanization of metal delivery to benches and established schedules and velocities of drawing.

There is a chapter on the quality control of wire. The school recommended the mechanization and automation of the quality control operations. The marking, lubrication and packing of the finished product at different plants are discussed and recommendations are made on the improvement of these operations and on the remuneration of OTK (Department of Technical Control) personnel.

This booklet can serve as a valuable manual for operators and technologists in wire drawing plants.

TREATMENT AND OVERHAUL OF ROLLING ROLLS

V.I. Merzlakov

Moscow, Metallurgizdat, 1957, 224 pp.

This book discusses the problems connected with the machining of rolls.

The first part of the book "General problems of roll machining" gives an account of rolls and the lathes for their machining. The author describes the roll turning tools and measuring and controlling instruments used in the machining of rolls. Instructions are given here also on how to use the drawings of rolling rolls.

A detailed description of the machining process is given in the second part of the book "Technology of rolling roll treatment". The preliminary machining of roll centers and the milling of roll wobblers are discussed. The technology of the machining of new rolls in stays on the shaping roll lathes and the reconditioning of worn rolls are described. A separate description of roll grinding after lathe machining is given.

The basic safety rules and the organization of work in the roll machining shops are discussed in the third section of the book. An account is given of the rules relating to lathe maintenance and the organization of the lathe operator's workshop.

The book aims at widening the theoretical knowledge of foreman and skilled operators and can also be of use to other workers in the steel rolling industry and to the undergraduate specializing in the field of metal working.

ROLLING OF VERY THIN STRIP

A.V. Tretyakov

Moscow, Metallurgizdat, 1957, 97 pp.

In this book the rolling of 0.05–0.15 mm thick strip is discussed. The author discusses the design of Soviet and foreign multi-roll stands for rolling this strip, giving a theoretical basis of some features of thin strip rolling.

The technology of rolling – reduction schemes, strip pickling and the modification of its mechanical properties depending on the thickness reduction – is described in full detail. The methods of thermal treatment of the strip and the design of heat-treating furnaces are also described.

The specifications of work rolls and backing up rolls, and the lubrication and cooling systems of the rolls are described. The types of faults in production and their causes are considered in detail. Considerable space is allotted to the strip thickness controlling instruments – contact (micrometers) and as well as contactless ones.

Information indispensable for the designing of multi-roll stands is contained in the chapters "Pressure of the metal on the rolls" and "Torque transmitted to the rolls".

The final chapter "Operating of cold rolling mills" sets out a method for evaluating the operating efficiency of mills.

The book is intended for technologists, designers, engineers and technicians in metallurgical and machine works.

The current Collection (No. 71) contains over 50 suggestions in various brances of metallurgy.

In the chapter "Raw material preparation and pig iron manufacture" attention is drawn to the proposal by V.P. Kanevsky (Krivorog Metallurgical Works) regarding the automation of the disc feeder starting-up. The adoption of this innovation resulted in improved uniform mixing of the ore for the sintering process.

A group of workers at the same plant suggested replacing the wooden lances for the cleaning of casting machine runners, with metallic tubes with cast iron tips. Such tubes are much more convenient in use than the wooden lances which burn quickly.

The workers of the Magnitogorsk Metallurgical Combine suggested mechanizing the preparation of lime solution for spraying slag cups.

In the chapter "Steel making", is described the design, developed by F.P. Burlyaev at the Dzerzhzinsky Works, of a machine for cutting the mold end plates. The efficiency of these machines is 1.8 times higher than of the old ones.

On the suggestion of I.I. Prokhorov the vertical gas ducts of open-hearth furnaces at "Azovstal" Works are provided with inspection holes. Thus it is possible to regularly inspect and clean the furnace bottom, the work of the operators being facilitated.

A new design of drying furnaces developed by G.S. Skrob, A.E. Pribytkov and N.K. Korolev ensures an eight fold reduction in the time of drying and an improvement of the drying process.

Many proposals are described in the chapter "Rolled production". The improvement of the ingot bogie proposed by A.I. Kalinin and L.P. Shcherbakov (Nizhne-Tagil Metallurgical Combine) deserves special attention. The conversion of the bogie from sliding to rolling bearings extended its life and made it possible to operate it at a greater speed.

As a result of the installation of an oil heater at the Zlatoust Metallurgical Works, as proposed by B.I. Chikin, and hence the use of heated oil (to 80°C) in the furnace the automatic operation of the furnace improved considerably.

The innovators at this Works, P.A. Korneev and M.E. Gluiberzon proposed to make the body of the working floor of the soaking pits from blooms instead of T-girders. By the adoption of this proposal the tilting of ingots in the soaking pits is greatly facilitated.

In the same chapter there are descriptions of a new design of a machine for fitting the billets for the anvil of the steam hammer, developed by G.D. Straty and V.V. Cherepov at the Taganrog Metallurgical Works; a modified design of the flue valve of the soaking pits; new designs for trolley cranes and electric current collectors and many other innovations, whose adoption facilitates the work of operators and will result in substantial savings.

Improvements in the design of some parts of the machines for tube manufacture are described in the chapter "Manufacture of tubular products". The chapter "Protecting metal industry" contains descriptions of a device for wire lubrication, new method of bending tongs production and a modified design of the jacket for a grinding machine.

A number of proposals are contained in the chapters "Blast furnace industry", "Power engineering" and "Miscellaneous".

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SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY
ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosenergoizdat	State Power Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LEIIZhT	Leningrad Power Inst. of Railroad Engineering
LET	Leningrad Elec. Engr. School
LETI	Leningrad Electrotechnical Inst.
LETIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MEP	Ministry of Electrical Industry
MES	Ministry of Electrical Power Plants
MESEP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTI	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOE	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIEL	Central Scientific Research Elec. Engr. Lab.
TsNIEL-MES	Central Scientific Research Elec. Engr. Lab.– Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIESKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZEI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. – Publisher.